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16. ABSTRACT

This document presents the findings of an extensive study of the ore mining and dressing industry, for the purpose of developing effluent limitations guidelines for existing point sources and standards for performance new sources, to implement Sections 304, and 307 of the Federal Water Pollution Control Act, as amended in 1977 by the Clean Water Act, P.L. 95-217.

Effluent limitations guidelines are set forth for the degree of effluent reduction attainable through the application of the best conventional pollutant control technology (BCT) and the degree of effluent reduction attainable through the application of the best available technology economically achievable BAT which must be achieved by existing point source sources. The standards of performance for new sources are set for the degree of effluent reduction which is achievable through application of the best available demonstrated control technology processes, operating methods, or other alternatives.

Supporting data and rationale for development of the proposed effluent limitation guidelines and standards of performance are contained in this report.

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DEVELOPMENT DOCUMENT
FOR PROPOSED
EFFLUENT LIMITATIONS GUIDELINES AND
NEW SOURCE PERFORMANCE STANDARDS
FOR THE
ORE MINING AND DRESSING
POINT SOURCE CATEGORY

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SECTION I

EXECUTIVE SUMMARY

This development document presents the technical data base developed by the EPA to support effluent limitations guidelines for the Ore Mining and Dressing Point Source Category. The Clean Water Act of 1977 sets forth various levels of technology to achieve these limitations. They are defined as best available technology economically achievable (BAT), best conventional pollutant control technology (BCT), and best available demonstrated technology (BADT). Effluent limitations guidelines based on the application of BAT and BCT are to be achieved by 1 July 1984. New source performance standards (NSPS) based on BADT are to be achieved by new facilities. These effluent limitations guidelines and standards are required by Sections 301, 304, 306, 307, and 501 of the Clean Water Act of 1977 (P.L. 95-217). They augment the interim final regulations based on BPT, which were first proposed on 6 November 1975. After extensive judicial review, the final BPT regulations were published on 11 July 1978 and sustained by the 10th Circuit Court of Appeals on 10 December 1979.

Although the Clean Water Act of 1977 established the primary legal framework for proposal of these limitations, EPA has also been guided by a series of legally-binding judicial actions. These include a series of settlement agreements, etc. into which EPA entered with the National Resources Defense Council (NRDC) and other environmental groups. The latest of these is NRDC v. Train, 8 ERC 2120 (D.D.C. 1976), modified, 12 ERC 1833 (D.D.C. 1979), aff'd and remd'd, EDF v. Costle, 14 ERC 2161 (D.D.C. 1980). The settlement agreement outlines a strategy for regulation of toxic pollutant discharges according to a schedule running through 1981 for 65 designated pollutant classes in 21 major industries, one of which is Ore Mining and Dressing. For the purpose of regulation, the list of 65 pollutant classes evolved into a list of 129 specific pollutants called "priority pollutants" because of their importance of controlling discharges of these toxic compounds. The priority pollutants serve as basis for EPA's development of effluent limitations based on BAT and BADT.

At present there are over 500 known major active ore mines (total operations may number as many as 1000) and over 150 active ore milling operations in the United States. Approximately two-thirds of these mines and mills are existing point source dischargers. The remainder do not discharge any process water. There are no known existing indirect dischargers and no new source indirect dischargers are anticipated. (Indirect dischargers are those facilities which discharge to a publicly owned treatment works.) Consequently, pretreatment standards, which control the level of pollutants which may be discharged from an industrial plant to a publicly owned treatment works, are not being proposed.

To recognize inherent differences in the industrial category, EPA established subcategories within the larger category. The BPT regulation for the ore mining and milling industry was divided into 7 major subcategories based upon metal ore and 21 subdivisions based upon whether the facility was a mine or mill and then further based upon the process employed at the mill. The BPT subcategorization is retained under BAT with one modification. The Ferroalloy ores subcategory which included tungsten and molybdenum ore mines and mills has been split apart. Molybdenum ore mines and mills are moved to the subcategory that already includes copper, lead, zinc, gold, silver, and platinum ore mines and mills. This new subcategory is renamed as the copper, lead, zinc, gold, silver, platinum, and molybdenum ores subcategory. Tungsten ore mines and mills are placed in a new and separate subcategory. Three new subcategories have been added since the time the court sustained the final BPT rule. These are to apply to BAT, BCT, and NSPS. Each of the three new subcategories consists of a single facility.

An extensive sampling and analysis effort was undertaken in 1977 and extends to the present. As part of this effort, 20 facilities were visited under screening and 14 facilities under verification sampling, six facilities were visited for solid waste and wastewater sampling, 12 treatability studies were performed at nine sites, and data collected by EPA Regions VI, VII, VIII, and X were reviewed to identify available treatment technologies and to determine effluent levels that could be achieved by these technologies. Six facilities were visited to collect cost information as well as wastewater samples. Four separate studies were performed by EPA's Industrial Environmental Research Laboratory in Cincinnati on the treatability of antimony, treatment alternatives for uranium mills, alternative flotation reagents to replace cyanide compounds, and the precision and accuracy of the analytical method for cyanide. The data base also includes the BPT record, National Pollutant Discharge Elimination System (NPDES) monitoring records, and data submitted by the industry.

Three studies have been performed to determine the cost of implementation of the candidate technologies. The first exercise determined the cost of technologies based on model (typical) facilities. The second costs the technologies in 1976 dollars based on actual data from approximately 90 mines and mills which had replied to an economic survey. These costs were verified in a third study since the industry is so economically sensitive. The costs presented in this document have been adjusted to 1979 dollars with appropriate inflation factors.

Executive Order 12291 (46 FR 13193-13198) requires that EPA and other agencies perform Regulatory Impact Analyses of major regulations. The three conditions that determine whether a regulation is classified as major are:

1. An annual effect on the economy of $100 million or more;

2. A major increase in costs or prices for consumers, individual industries, federal, state, or local government agencies, or geographic regions; or

3. Significant adverse effects on competition, employment, investement productivity, innovation, or on the ability of United States based enterprises to compete with foreign based enterprises in domestic or export markets.

Under Executive Order 12291, EPA must judge whether a regulation is "major" and therefore subject to the requirement of a Regulatory Impact Analysis. This regulation is not major and does not require a Regulatory Impact Analysis because the annual effect on the economy is less than $100 million, it will not cause a mjaor increase in costs, or significant adverse effects on the industry.

This regulation was submitted to the Office of Management and Budget for review as required by Executive Order 12251. Any comments from OMB and EPA's responses to those comments are available for public inspection at the EPA Public Information Reference Unit, Room 2922 (EPA Library), Environmental Protection Agency, 401 M Street, S.W., Washington, D.C.

BEST AVAILABLE TECHNOLOGY ECONOMICALLY ACHIEVABLE (BAT)

The presence or absence of the 129 toxic pollutants and several conventional and nonconventional pollutants has been determined as a result of the sampling and analysis program. One hundred twenty-four of the 129 toxic pollutants have been excluded in all subcategories based upon criteria contained in the Settlement Agreement cited previously: (1) they were not detected, (2) they were present at levels not treatable by known technologies, or (3) they were effectively controlled by technologies upon which other effluent limitations are based. The five remaining toxics were excluded in some individual subcategories. Where specific toxic pollutants are to be controlled with effluent limitations, i.e., they were not excluded from the entire category or individual subcategories, effluent limitations for those pollutants are proposed. A number of end-of-pipe treatment alternatives were considered for BAT, but were reduced to three alternatives: (1) secondary settling; (2) flocculation/coagulation; and (3) granular media filtration. The remaining alternatives were eliminated because of high costs and because some technologies were not applicable to an industrial discharge characterized by extremely high flows and comparatively low concentrations of pollutants in treated effluents. The three options considered for controlling toxic metals were "add on" to

BPT facilities which consist of lime precipitation and settling. Of the three alternatives, no statistically significant differences were discerned among the effluents from these technologies.

Of these alternatives, secondary settling would require the least expenditure. A statistical analysis of plant data from facilities using secondary settling was used to derive achievable levels which are more stringent than BPT. However, based on the following considerations, the Agency has determined that nationally applicable regulations based on secondary settling are not warranted. First, in each subcategory, at least 95 percent of the relevant pollutants are removed by BPT. Those pollutants remaining are generally sulfide and oxide compounds in the form of ore and gangue. Second, the Agency's environmental assessment concluded that for the industrial category, the only environmentally significant pollutants after stream flow dilution are cadmium and arsenic and there is no appreciable reduction of these between BPT and the derived levels. Finally, the BPT limitations in this industry are generally more stringent than BAT limitations being considered in other industries.

The BPT regulation provides for relief from effluent limitations, including zero discharge, during periods of precipitation. The basis of the precipitation exemption is that treatment facilities must be designed, constructed, and maintained to include the volume of water that would result from a 10-year, 24-hour precipitation event. The same storm provision is retained in BAT.

Where BPT is zero discharge for a subcategory, BAT is also zero discharge. In subcategories where toxic pollutants were found, BAT effluent limitations are proposed at BPT levels. As stated previously all but five toxic pollutants are excluded from regulation. These five are cadmium, copper, lead, mercury and zinc.

BAT effluent limitations are not being established for asbestos. BPT and BCT effluent limitations for TSS will effectively control the discharge of asbestos. Available data demonstrated that, as TSS levels are reduced in wastewater from mines and mills, asbestos levels are reduced concomitantly, although the reduction can not be quantified precisely. However, when TSS is reduced to less than or equal to 30 mg/l the data indicate that asbestos is reduced to levels near observed background concentrations (roughly 10^8 fibers per liter).

Uranium mills are excluded from BAT because pollutants from the subcategory are from a single source and are uniquely related to that source.

Cyanide is not regulated under BAT. A special study of the precision and accuracy of the method was performed as part of the

BAT review. Specific technology for the destruction of cyanide was considered, but is not necessary because in-process controls and retention of wastewater in tailing ponds reduce cyanide to less than 0.4 mg/l based on the precision and accuracy of the analytical method for wastewater discharges from ore mines and mills.

Gold placer mines are not regulated under the proposed BAT and the subpart is reserved. Almost all gold placer mines are located in remote areas of Alaska. No economic analysis has been performed on these placer mines because no data are available despite requests to industry for information. The Preamble to the Proposed Regulation requests specific effluent data and cost and cash flow data from palcer mines in order that an economic impact assessment can be made before the regulation is proposed.

NEW SOURCE PERFORMANCE STANDARDS (NSPS)

New facilities have an opportunity to implement the best and most efficient ore mining and milling processes and wastewater treatment technologies. Accordingly, Congress directed EPA to consider the best demonstrated process changes and end-of-pipe treatment technologies capable of reducing pollution to the maximum extent feasible through a standard of performance which includes, "where practicable, a standard permitting zero discharge of pollutants".

NSPS for uranium mills is proposed as zero discharge. Zero discharge is well demonstrated at existing uranium mills (18 of 19 do not discharge). New uranium mills in arid areas can achieve zero discharge as cheaply as they can install treatment to meet BPT limitations. Arid areas are those in which the volume of water evaporated exceeds the volume resulting from precipitation. The Agency knows of no mills actually planned for humid areas.

NSPS for froth-flotation mills is proposed as zero discharge. Zero discharge, based on total impoundment and recycle, or evaporation, or a combination of these technologies, is demonstrated as practicable at 46 of the 90 existing mills using froth flotation for which EPA has data. Zero discharge (based on recycle) was rejected as BAT because of the cost of retrofitting the process in some existing mills and the potential changes required in some existing mill processes where two or more concentrates are recovered from the raw ore. The Agency's data indicates that new source froth flotation mills can achieve zero discharge as cheaply as they can install treatment to meet BPT.

The storm provision would depend on whether a facility is subject to zero discharge. For NSPS requiring zero discharge, the excursion applies only when a 10-year, 24-hour or greater storm occurs and for NSPS allowing discharge subject to limitations, the excursion will continue to be tied to the design criteria.

BCT EFFLUENT LIMITATIONS

BCT was not intended to act as another limitation; rather, it replaces BPT for control of the conventional pollutants: total suspended solids (TSS), pH, biochemical oxygen demand (BOD), oil and grease (O&G), and fecal coliform. Fecal coliform, BOD, and O&G are not found in significant concentrations in this industry. TSS and pH are central to control and treatment of the toxic metals and are limited under BPT.

Since BCT is equivalent to BPT, no cost is implied to meet BCT effluent limitations. BCT would pass any cost test since there is no cost to implement BCT. The Agency is currently developing a new BCT cost methodology. It is possible, though unlikely, that a treatment technology more stringent than BPT will provide additional removal of conventional pollutants and pass the new cost test, when developed. In that event, the Agency will propose new BCT limitations.

In general, ore mines and mills are located in rural areas, far from a POTW.

The 1977 Amendments added Section 301(b)(2)(E) to the Act establishing "best conventional pollutant control technology" (BCT) for discharges of conventional pollutants from existing industrial point sources. Conventional pollutants are those defined in Section 304(a)(4) [biological oxygen demanding pollutants (BOD5), total suspended solids (TSS), fecal coliform, and pH], and any additional pollutants defined by the Administrator as "conventional" [oil and grease, 44 FR 44501, July 30, 1979].

BCT is not an additional limitation but replaces BPT for the control of conventional pollutants. In addition to other factors specified in section 304(b)(4)(B), the Act requires that BCT limitations be assessed in light of a two part "cost-reasonableness" test. American Paper Institute v. EPA, 660 F.2d 954 (4th Cir. 1981). The first test compares the cost for private industry to reduce its conventional pollutants with the costs to publicly owned treatment works for similar levels of reduction in their discharge of these pollutants. The second test examines the cost-effectiveness of additional industrial treatment beyond BPT. EPA must find that limitations are "reasonable" under both tests before establishing them as BCT. In no case may BCT be less stringent than BPT.

EPA publisyed its methodology for carrying out the BCT analysis on August 29, 1979 (44 FR 50732). In the case mentioned above, the Court of Appeals ordered EPA to correct data errors underlying EPA's calculation of the first test, and to apply the second cost test. (EPA had argued that a second cost test was not required.)

While EPA has not yet proposed or promulgated a revised BCT methodology in response to the American Paper Institute v. EPA decision mentioned earlier, EPA is proposing BCT limitations for this category. These limits would be identical to those for BPT. As BPT is the minimal level of control required by law, no possible application of the BCT cost tests could result in BCT limitations lower than those proposed today. Accordingly, there is no need to wait until EPA revises the BCT methodology before proposing BCT limitations.

Prior EPA Regulations

On 6 November 1975, EPA published interim final regulations establishing BPT requirements for existing sources in the ore mining and dressing industry (see 40 FR 51722). These regulations became effective upon publication. However, concurrent with their publications, EPA solicited public comments with a view to possible revisions. On the same date, EPA also published proposed BAT, NSPS, and pretreatment standards for this industry

SECTION II

INTRODUCTION

PURPOSE

This study determined the presence and concentrations of the 129 toxic or "priority" pollutants in the ore mining and dressing point source category for possible regulation. This development document presents the technical data base compiled by EPA with regard to these pollutants and their treatability for regulation under the Clean Water Act. The concentrations of conventional and nonconventional pollutants were also examined for the establishment of effluent limitations guidelines. Treatment technologies were also assessed for designation as the best available demonstrated technology (BADT) upon which new source performance standards (NSPS) are based. This document outlines the technology options considered and the rationale for selecting each technology level. These technology levels are the basis for the proposed regulation effluent limitations.

LEGAL AUTHORITY

The regulations are proposed under authority of Sections 301, 304, 306, 307, 308, and 501 of the Clean Water Act (the Federal Water Pollution Control Act Amendments of 1972, 33 USC 1251 et seq., as amended by the Clean Water Act of 1977, P.L. 95-217) (the "Act"). These regulations are also proposed in response to the Settlement Agreement in Natural Resources Defense Council, Inc., v. Train, 8 ERC 2120 (D.D.C. 1976), modified, 12 ERC 1833 (D.D.C. 1979).

The Clean Water Act

The Federal Water Pollution Control Act Amendments of 1972 established a comprehensive program to "restore and maintain the chemical, physical, and biological integrity of the Nation's waters," Section 101(a). By 1 July 1977, existing industrial dischargers were required to achieve "effluent limitations requiring the application of the best practicable control technology currently available" (BPT), Section 301(b)(1)(A). By 1 July 1983, these dischargers were required to achieve "effluent limitations requiring the application of the best available technology economically achievable . . . which will result in reasonable further progress toward the national goal of eliminating the discharge of all pollutants" (BAT), Section 301(b)(2)(A). New industrial direct dischargers were required to comply with Section 306 new source performance standards (NSPS), based on best available demonstrated technology. The requirements for direct dischargers were to be incorporated into National Pollutant Discharge Elimination System (NPDES) permits issued under Section 402 of the Act. Although Section 402(a)(1) of the 1972 Act authorized the setting of requirements for direct dischargers on a case-by-case basis, Congress intended that for the most part, control requirements would be based on regulations promulgated by the Administrator of EPA. Section 304(b) of the Act required the Administrator to promulgate regulations providing guidelines for effluent limitations setting forth the degree of effluent reduction attainable through the application of BPT and BAT. Moreover, Sections 304(c) and 306 of the Act required promulgation of regulations for NSPS. In addition to these regulations for designated industry categories, Section 307(a) of the Act required the Administrator to promulgate effluent standards applicable to all dischargers of toxic pollutants. Finally, Section 501(a) of the Act authorized the Administrator to prescribe any additional regulations "necessary to carry out his functions" under the Act. EPA was unable to promulgate many of these regulations by the dates contained in the Act. In 1976, EPA was sued by several environmental groups, and in settlement of this lawsuit EPA and the plaintiffs executed a Settlement Agreement which was approved by the Court. This Agreement required EPA to develop a program and adhere to a schedule for promulgating BAT effluent limitations guidelines, and new source performance standards covering 65 classes of toxic pollutants (subsequently defined by the Agency as 129 specific "priority pollutants") for 21 major industries. See Natural Resources Defense Council, Inc. v. Train, 8 ERC 2120 (D.D.C. 1976), modified, 12 ERC 1833 (D.D.C. 1979).

On 27 December 1977, the President signed into law the Clean Water Act of 1977 ("the Act"). Although this law makes several important changes in the Federal Water Pollution Control Program, its most significant feature is its incorporation of several basic elements of the Settlement Agreement program for toxic pollution control. Sections 301(b)(2)(A) and 301(b)(2)(C) of the Act now require the achievement, by 1 July 1984, of the effluent limitations requiring application of BAT for toxic pollutants, including the 65 priority pollutants and classes of pollutants that Congress declared toxic under Section 307(a) of the Act. Likewise, EPA's programs for new source performance standards are now aimed principally at toxic pollutant controls. Moreover, to strengthen the toxics control program, Section 304(e) of the Act authorizes the Administrator to prescribe "best management practices" (BMPs) to control the release of toxic and hazardous pollutants from plant site runoff; spillage or leaks; sludge or waste disposal; and drainage from raw material storage associated with, or ancillary to, the manufacturing or treatment process.

The proposed regulations provide effluent limitations guidelines for BAT and establish NSPS on the basis of the authority granted in Sections 301, 304, 306, 307, and 501 of the Clean Water Act. Pretreatment Standards (PSES and PSNS) are not proposed for the ore mining and dressing category since no known indirect dischargers exist nor are any known to be in the planning stage.

(see 40 FR 51738). Comments were also solicited on these proposals.

On 24 May 1976, as a result of the public comments received, EPA suspended certain portions of the interim final BPT regulations and solicited additional comments (see 41 FR 21191). EPA promulgated revised, final BPT regulations for the ore mining and dressing industry on 11 July 1978, (see 43 FR 29711, 40 CFR Part 440). On 8 February 1979, EPA published a clarification of the regulations as they apply to storm runoff (see 44 FR 7953). On 1 March 1979, the Agency amended the final regulations by deleting the requirements for cyanide applicable to froth flotation mills in the base and precious metals subcategory (see 44 FR 11546).

On 10 December 1979, the United States Court of Appeals for the Tenth Circuit upheld the BPT regulations, rejecting challenges brought by five industrial petitioners, Kennecott Copper Corp. v. EPA, 612 F.2d 1232 (10th Cir. 1979). The Agency withdrew the proposed BAT, NSPS, and pretreatment standards on 19 March 1981 (see 46 FR 17567).

Industry Overview

The ore mining and dressing industry is both large and diverse. It includes the ores of 23 separate metals and is segregated by the U.S. Bureau of the Census Standard Industrial Classification (SIC) into nine major codes: SIC 1011, Iron Ore; SIC 1021, Copper Ores; SIC 1031, Lead and Zinc Ores; SIC 1041, Gold Ores; SIC 1044, Silver Ores; SIC 1051, Aluminum Ore; SIC 1061, Ferroalloy Ores including Tungsten, Nickel, and Molybdenum; SIC 1092 Mercury Ores; SIC 1094, Uranium, Radium, and Vanadium Ores; and SIC 1099, Metal Ores, Not Elsewhere Classified including Titanium and Antimony. Over 500 active mining and over 150 milling operations are located in the United States. Many are in remote areas. The industry includes facilities that mine ores to produce metallic products and all ore dressing and beneficiating operations at mills operated either in conjunction with a mine operation or at a separate location.

Summary of Methodology

From 1973 through 1976, EPA emphasized the achievement of limitations based on application of best practicable technology (BPT) by 1 July 1977. In general, this technology level represented the average of the best existing performances of well-known technologies for control of familiar pollutants associated with the industry. In this industry, many metal pollutants that Congress subsequently designated as toxic were also regulated under BPT.

This rulemaking ensures the achievement, by 1 July 1984, of limitations based on application of the best available technology economically achievable (BAT). In general, this technology level represents the best economically achievable performance in any industry category or subcategory. Moreover, as a result of the Clean Water Act of 1977, the emphasis of EPA's program has shifted from control of "classical" pollutants to the control of toxic substances.

EPA's implementation of the Act is described in this section and succeeding sections of this document. Initially, because in many cases no public or private agency had done so, EPA, its laboratories, and consultants had to develop analytical methods for toxic pollutant detection and measurement. EPA then gathered technical and economic data about the industry. A number of steps were involved in arriving at the proposed limitations.

First, EPA studied the ore mining and dressing industry to determine whether differences in raw materials; final products; manufacturing processes; equipment, age, and size of plants; water usage; wastewater constituents; or other factors required the development of separate effluent limitations and standards for different subcategories and segments of the industry. This study included identifying raw waste and treated effluent characteristics, including: the sources and volume of water used, the processes employed, and the sources of pollutants and wastewater in the plant and the constituents of wastewater, including toxic pollutants. EPA then identified the constituents of wastewaters that should be considered for effluent limitations guidelines and standards of performance.

Next, EPA identified several distinct control and treatment technologies, including both in-plant and end-of-process technologies, that are in use or capable of being used in the ore mining and dressing industry. The Agency compiled and analyzed historical and newly generated data on the effluent quality resulting from the application of these technologies. The long-term performance, operational limitations, and reliability of each treatment and control technology were also identified. In addition, EPA considered the non-water quality environmental impacts of these technologies, including impacts on air quality, solid waste generation, water availability, and energy requirements.

The Agency then estimated the costs of each control and treatment technology from unit cost curves developed by standard engineering analyses as applied to ore mining and dressing wastewater characteristics. EPA derived unit process costs from representative plant characteristics (production and flow) applied to each treatment process (i.e., secondary settling, pH adjustment and settling, granular-media filtration, etc.). These unit process costs were added to yield total cost at each treatment level. After confirming the reasonableness of this methodology by comparing EPA cost estimates with treatment system costs supplied by the industry, the Agency evaluated the economic impacts of these costs.

After considering these factors, EPA identified various control and treatment technologies as BAT and NSPS. The proposed regulation, however, does not require the installation of any particular technology or limit the choices of technologies that may be used in specific situations. Rather, it requires achievement of effluent limitations that represent the proper design, construction, and operation of these or equivalent technologies.

The effluent limitations for ore mining and dressing BAT, BCT, and NSPS are expressed in concentrations (e.g., milligrams of pollutant per liter of wastewater) rather than loading per unit(s) of production (e.g., kg of pollutant per metric ton of product) because correlating units of production and wastewater discharged by mines and mills was not possible for this category. The reasons are:

1. The quantity of mine water discharged varies considerably from mine to mine and is influenced by topography, climate, geology (affecting infiltration rates) and the continuous nature of water infiltration regardless of production rates. Mine water may be generated and required to be treated and discharged even if production is reduced or terminated.

2. Consistent water use and loss relationships for ore mills could not be derived from facility to facility within a subcategory because of wide variations in application of specific processes. The subtle differences in ore mineralogy and process development may require the use of differing amounts of water and process reagents but do not necessarily require different wastewater treatment technology(ies).

The Agency is not proposing pretreatment standards because it does not know of any existing facilities that discharge to POTWs or any that are planned.

Data Gathering Efforts

Data gathering for the ore mining and dressing industry included an extensive collection of information:

1. Screening and verification sampling and analysis programs
2. Engineereing cost site visits
3. Supporting data from EPA regional offices
4. Treatability studies
5. Industry self-monitoring sampling
6. BPT data base

7. Placer study

8. Titanium sand dredges study

9. Uranium study

10. Solid waste study

EPA began an extensive data collection effort during 1974 and 1975 to develop BPT effluent standards. These data included results from sampling programs conducted by the Agency at mines and mills and an assimilation of historical data supplied by the industry, the Bureau of Mines, and other sources. This information characterized wastewaters from ore mining and milling operations according to what were then considered key parameters-total suspended solids, pH, lead, zinc, copper, and other metals. However, little information on other environmental parameters, such as other toxic metals and organics, was available from industry or government sources. To establish the levels of these pollutants, the Agency instituted a second sampling and analysis program to specifically address these toxic substances, including 129 specific toxic pollutants for which regulation was mandated by the Clean Water Act.

EPA began the second sampling and analysis program (screening and verification sampling) in 1977 to establish the quantities of toxic, conventional, and nonconventional pollutants in ore mine drainage and mill processing effluents. EPA visited 20 and 14 facilities respectively for screening and verification sampling.

EPA selected at least one facility in each major BPT subcategory. The sites selected were representative of the operations and wastewater characteristics present in particular subcategories. These facilities were visited from April through November 1977. To determine these sites, the agency reviewed the BPT data base and industry as a whole, with consideration to:

1. Those using reagents or reagent constituents on the toxic pollutants list;

2. Those using effective treatment for BPT regulated pollutants;

3. Those for which historical data were available as a means of verifying results obtained during screening; and

4. Those suspected of producing wastewater streams that contain pollutants not traditionally monitored.

After reviewing screen sampling analytical results, EPA selected 14 sites for verification sampling visits. Because most of the organic toxic pollutants were either not detected or detected only at low concentrations in the screen samples, the Agency emphasized verification sampling for total phenolics (4AAP), total cyanide, asbestos (chrysotile), and toxic metals.

EPA revisited six of the facilities to collect additional data on concentrations of total phenolics (4AAP), total cyanide, asbestos (chrysotile), and to confirm earlier measurements of these parameters.

After completing verification sampling, EPA conducted sampling of two additional sites. At one molybdenum mill operation, a complete screen sampling effort was performed to determine the presence of toxic pollutants and to collect data on the performance of a newly installed treatment system. The second facility, a uranium mine/mill, was sampled to collect data on a facility removing radium 226 by ion exchange. Samples collected at this facility were not analyzed for organic toxic pollutants.

The Agency conducted a separate sampling effort to evaluate treatment technologies at Alaskan placer gold mines. This study was undertaken because gold placer mining was reserved under BPT rulemaking and because little data were previously available on the performance of existing treatment systems.

Industrial self-sampling was conducted at three facilities visited during screen sampling to supplement and expand the data for these facilities. The programs lasted from two to twelve weeks. EPA selected two operations because they had been identified during the BPT study as two of the best treatment facilities; the third because additional data on long-term variations in waste stream charactersitics at these sites were needed to supplement the historical discharge monitoring data, to reflect any recent changes or improvements in the treatment technology used, and to confirm that variations in raw wastewater levels did not affect concentrations in treated effluents.

The Agency's regional surveillance and analysis groups performed additional sampling at 14 facilities: nine in Colorado, Idaho, Wyoming, and Montana; one in Arkansas; and four in Missouri.

Discharge monitoring reports were collected from EPA regional offices for many of the ore producing facilities with treatment systems. These data were used in evaluating the variations in flow and wastewater characteristics associated with mine drainage and mill wastewater.

The Agency took samples during the cost-site visits, although the primary reason for the visits was to collect data that would assist the Agency in developing unit process cost curves and that would verify the cost assumption made. However, since many of the sites had been sampled previously, the new sampling data obtained served as additional verification of waste characterization data.

EPA conducted 13 treatability studies to characterize performance of alternative treatment technologies on ore mine and mill wastewaters. Secondary settling, flocculation, granular media filtration, ozonation, alkaline chlorination and hydrogen peroxide treatment were all examined in bench- and pilot-scale studies. The data obtained from these studies were compared with data obtained on the performance of these systems in actual operations on pilot and full scale. In addition, the data were used to determine the range of variability that might be expected for these technologies, especially during periods of steady running.

EPA obtained the data for its economic analysis primarily from a survey conducted under Section 308 of the Clean Water Act. The Agency sent questionnaires to 138 companies engaged in mining and milling of metal ores. The data collected included production levels, employment, revenue, operating costs, working capital, ore grade, and other relevant information. The economic survey data were supplemented by data from government publications, trade journals, and visits to several mine/mills.

SECTION III

INDUSTRY PROFILE

ORE BENEFICIATION PROCESSES

As mined, most ores contain the valuable metals (values) disseminated in a matrix of less valuable rock (gangue). The purpose of ore beneficiation is the separation of the metal bearing minerals from the gangue to yield a product which is higher in metal content. To accomplish this, the ore must generally be crushed and/or ground small enough so that each particle contains mostly the mineral to be recovered or mostly gangue. The separation of the particles on the basis of some difference between the ore mineral and the gangue can then yield a concentrate high in metal value, as well as waste rock (tailings) containing very little metal. The separation is never perfect, and the degree of success which is attained is generally described by two numbers: (1) percent recovery and (2) grade of the product (usually a concentrate). Widely varying results are obtained in beneficiating different ores; recoveries may range from 60 percent or less to greater than 95 percent. Similarly, concentrates may contain less than 60 percent or more than 95 percent of the primary ore mineral. In general, for a given ore and process, concentrate grade and recovery are inversely related. Higher recovery is achieved only by including more gangue, thereby yielding a lower grade concentrate. The process must be optimized, trading off recovery against the value (and marketability) of the concentrate produced. Depending on end use, a particular grade of concentrate is desired, and specific gangue components are limited as undesirable impurities.

Many properties are used as the basis for separating valuable minerals from gangue, including: specific gravity, conductivity, magnetic permeability, affinity for certain chemicals, solubility, and the tendency to form chemical complexes. Processes for effecting the separation may be generally considered as:

1. gravity concentration
2. magnetic separation
3. electrostatic separation
4. flotation
5. leaching

Amalgamation and cyanidation are variants of the leaching process. Solvent extraction and ion exchange are widely applied techniques for concentrating metals from leaching solutions and for separating them from dissolved contaminants.

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These processes are discussed in general terms in the paragraphs which follow. This discussion is not meant to be all inclusive; rather, it is to discuss the primary processes in current use in the ore mining and milling industry. Details of some processes used in typical mining and milling operations have been discussed including presentation of process flowcharts, in Appendix A, Industry Processes.

Gravity Concentration Processes

Gravity concentration processes exploit differences in density to separate valuable ore minerals from gangue. Several techniques (jigging, tabling, spirals, sink/float separation, etc.) are used to achieve the separation. Each is effective over a somewhat limited range of particle sizes, the upper bound of which is set by the size of the apparatus and the need to transport ore within it, and the lower bound by the point at which viscous forces predominate over gravity and render the separation ineffective. Selection of a particular gravity based process for a given ore will be strongly influenced by the size to which the ore must be crushed or ground to separate values from gangue as well as by the density difference and other factors.

Most gravity techniques depend on viscous forces to suspend and transport gangue away from the heavier valuable mineral. Since the drag forces on a particle depend on its area, and its weight depends on its volume, particle size as well as density will have a strong influence on the movement of a particle in a gravity separator. Smaller particles of ore mineral may be carried with the gangue despite their higher density, or larger particles of gangue may be included in the gravity concentrate. Efficient separation, therefore, requires a process feed with uniform particle sizes. A variety of classifiers (spiral and rake classifiers, screens, and cyclones) are used to assure a reasonably uniform feed. At some mills, a number of sized fractions of ore are processed in different gravity separation units.

Viscous forces on the particles set a lower particle size limit for effective gravity separation by any technique. For very small particles, even slight turbulence may suspend the particle for long periods of time, regardless of density. Such suspensions (slimes), cannot be recovered by gravity techniques and may cause very low recoveries in gravity processing of friable ores, such as scheelite (calcium tungstate, $CaWO_4$).

Jigs

Jigs of many different designs are used to achieve gravity separation of relatively coarse ore usually between 0.5 mm (0.02 inch) and 25 mm (1 inch) in diameter. In general, ore is fed as a thick slurry to a chamber in which agitation is provided by a pulsating plunger or other such mechanism. The feed separates into layers by density within the jig, the lighter gangue being drawn off at the top, with the water overflow and the denser mineral drawn off at a screen on the bottom. Often a bed of coarser ore or iron shot is used to aid the separation; the dense ore mineral migrates down through the bed under the influence of the agitation within the jig. Several jigs are often used in series to achieve both acceptable recovery and high concentrate grade. Jigs are employed in the processing of ores of iron, gold, and ferroalloys.

Tables

Shaking tables of a wide variety of designs have found widespread use as an effective means of achieving gravity separation of finer ore particles 0.08 mm (.003 inch) to 2.5 mm (0.1 inch) in diameters. Fundamentally, they are tables over which flow ore particles suspended in water. A series of ridges or riffles perpendicular to the water flow traps heavy particles while lighter ones are suspended and flow over the obstacles with the water stream. The heavy particles move along the ridges to the edge of the table and are collected as concentrate (heads), while the light material which follows the water flow is generally a waste stream (tails). Between these streams may be some material (middlings) which has been partially diverted by the riffles. These are often collected separately and returned to the table feed. Reprocessing of either heads or tails, or both, and multiple stage tabling are common. Tables may be used to separate minerals of minor density differences, but uniformity of feed becomes extremely important in such cases. Tables are employed in the processing of ores of gold, ferroalloys, titanium, and zirconium.

Spirals

Humphreys spiral separators, a relatively recent development, provide an efficient means of gravity separation for large volumes of material between 0.1 mm and 2 mm (.004 inch to .08 inch) in diameter. They have been widely applied, particularly, in the processing of heavy sands for ilmenite ($FeTiO_3$) and monazite (a rare earth phosphate). Spirals consist of a helical conduit (usually of five turns) about a vertical axis. A slurry of ore is fed to the conduit at the top and flows down the spiral under gravity. The heavy minerals concentrate along the inner edge of the spiral from which they may be withdrawn through a series of ports. Wash water may also be added through ports along the inner edge to improve the separation efficiency. A single spiral may typically be used to process 0.5 to 2.4 metric tons (0.55 to 2.64 short tons) of ore per hour; in large plants, as many as several hundred spirals may be run in parallel. Spirals are used for processing ores of ferroalloys, titanium, and zirconium.

Sink/Float Separation

Sink/float separators differ from most gravity methods in that bouyancy forces are used to separate the various minerals on the basis of density. The separation is achieved by feeding the ore to a tank containing a medium of higher density than the gangue and less than the valuable ore minerals. As a result, the gangue floats and overflows the separation chamber, and the denser values sink and are drawn off at the bottom by a bucket elevator or similar contrivance. Because the separation takes place in a relatively still basin and turbulence is minimized, effective separation may be achieved with a more heterogeneous feed than for most gravity separation techniques. Viscosity does, however, place a lower bound on separable particle size because small particles settle very slowly, limiting the rate at which ore may be fed. Further, very fine particles must be excluded since they mix with the separation medium, altering its density and viscosity.

Media commonly used for sink/float separation in the ore milling industry are suspensions of very fine ferrosilicon or galena (PbS) particles. Ferrosilicon particles may be used to achieve medium specific gravities as high as 3.5 and are used in heavy-medium separation. Galena, used in the "Huntington-Heberlein" process, allows the achievement of somewhat higher densities. The particles are maintained in suspension by a modest amount of agitation in the separator and are recovered for reuse by washing both values and gangue after separation.

Sink/float separation techniques are employed for processing ores of iron.

Magnetic Separation

Magnetic separation is widely applied in the ore milling industry, both for the extraction of values from ore and for the separation of different valuable minerals recovered from complex ores. Extensive use of magnetic separation is made in the processing of ores of iron, columbium, and tungsten. The separation is based on differences in magnetic permeability (which, although small, is measurable for almost all materials). This method is effective in handling materials not normally considered magnetic. The basic process involves the transport of ore through a region of high magnetic field gradient. The most magnetically permeable particles are attracted to a moving surface by a large electromagnet. The particles are carried out of the main stream of ore by the moving surface and as it leaves the high field region the particles drop off into a hopper or onto a conveyor leading to further processing.

For large scale applications (particularly in the iron ore industry) large, rotating drums surrounding the magnet are used. Although dry separators are used for rough separations, drum separators are most often run wet on the slurry produced in grinding mills. Where smaller amounts of material are handled, wet and crossed-belt separators are frequently employed.

Magnetic separation is used in the beneficiation of ores of iron, ferroalloys, titanium, and zirconium are discussed in Appendix A.

Electrostatic Separation

Electrostatic separation is used to separate minerals on the basis of their conductivity. It is an inherently dry process using very high voltages (typically 20,000 to 40,000 volts). In a typical implementation, ore is charged to 20,000 to 40,000 volts, and the charged particles are dropped onto a conductive rotating drum. The conductive particles discharge very rapidly, are thrown off, and collected. The nonconductive particles keep their charge and adhere by electrostatic attraction to be removed from the drum separately. Specific instances in which electrostatic separation has been used for processing ores of ferroalloys, titanium, and zirconium, are discussed in Appendix A.

Flotation Processes

Basically, flotation is a process where particles of one mineral or group of minerals are made by addition of chemicals to adhere preferentially to air bubbles. When air is forced through a slurry of mixed minerals, the rising bubbles carry with them the particles of the mineral(s) to be separated from the matrix. If a foaming agent is added which prevents the bubbles from bursting when they reach the surface, a layer of mineral laden foam is built up at the surface of the flotation cell which may be removed to recover the mineral. Requirements for the success of the operation are that particle size be small, that reagents be compatible with the mineral, and that water conditions in the cell not interfere with attachment of reagents to the mineral or to air bubbles.

Flotation concentration has become a mainstay of the ore milling industry. Because it is adaptable to very fine particle sizes (less than 0.001 cm), it allows high rates of recovery from slimes, which are inevitably generated in crushing and grinding and which are not generally amenable to physical processing. As a physio-chemical surface phenomenon, it can often be made highly specific, allowing production of high grade concentrates from relatively low grade ore (e.g., over 95 percent MoS_2 concentrate from 0.3 percent ore). Its specificity also allows separation of different ore minerals (e.g., CuS, PbS, and ZnS) and operation with minimum reagent consumption since reagent interaction is typically only with the particular materials to be floated or depressed.

Details of the flotation process (exact dosage of reagents, fineness of grinds, number of regrinds, cleaner flotation steps, etc.) differ at each operation where it is practiced and may often vary with time at a given mill. A complex system of reagents is generally used, including four basic types of compounds: collectors, frothers, activators, and depressants. Collectors serve to attach ore particles to air bubbles formed in the flotation cell. Frothers stablilize the bubbles to create a foam which may be effectively recovered from the water surface. Activators enhance the attachment of specific kinds of particles to the air bubbles, and depressants prevent it. Frequently, activators are used to allow flotation of ore which has been depressed in an earlier stage of the process. In almost all cases, use of each reagent in the mill is low (generally, less than 0.5 kg per ton of ore processed), and the bulk of the reagent adheres to tailings or concentrates.

Sulfide minerals are readily recovered by flotation using similar reagents in small doses; although reagent requirements vary throughout the class. Sulfide flotation is most often carried out at alkaline pH. Collectors are most often alkaline xanthates having two to five carbon atoms, for example, sodium ethyl xanthate ($NaS_2COC_2H_5$). Frothers are generally organics with a soluble group and a nonwettable hydrocarbon. Pine oil hydroxyl ($C_6H_{12}OH$), for example, is widely used to allow separate recovery of metal values from mixed sulfide ores. Sodium cyanide is widely used as a pyrite depressant. Activators useful in sulfide ore flotation may include cuprous sulfide and sodium sulfide. Sulfide minerals of copper, lead, zinc, molybdenum, silver, nickel, and cobalt are commonly recovered by flotation.

Many minerals in addition to sulfides may be, and often are, recovered by flotation. Oxidized ores of iron, copper, manganese, the rare earths, tungsten, titanium, columbium and tantalum, for example, may be processed in this way. Flotation of these ores involves a very different group of reagents from sulfide flotation and has, in some cases, required substantially larger dosages. These flotation processes may be more sensitive to feed water conditions than sulfide floats. They are less frequently run with recycled water or untreated water. Collector reagents used include fatty acids (such as oleic acid or soap skimmings), fuel oil, various amines, and compounds such as copper sulfate, acid dichromate, and sulfur dioxide as conditioners.

Flotation is also used to process ores of iron, copper, lead and zinc, gold, silver, ferroalloys, mercury, and titanium.

Leaching

Ores can be beneficiated by dissolving away either gangue or values in aqueous acids or bases, liquid metals, or other specific solutions. This process is called leaching.

Leaching solutions are categorized as strong, general solvents (e.g., acids) and weaker, specific solvents (e.g., cyanide). The acids dissolve any metals present, which often include gangue constituents (e.g., calcium from limestone). They are convenient to use since the ore does not have to be very finely ground, and separation of the tailings from the value bearing (pregnant) leach is then not difficult.

Specific solvents attack only one (or, at most, a few) ore constituent(s). Ore must be finely ground to expose the values. Heat, agitation, and pressure are often used to speed the actions of the leach, and it is difficult to separate the solids (often in the form of slimes) from the pregnant leach.

Countercurrent leaching, preneutralization of lime in the gangue, leaching in the grinding process, and other combinations of processes are often seen in the industry. The values contained in the pregnant leach solution are recovered by one of several methods, including precipitation (e.g., of metal hydroxides from acid leach by raising pH), electrowinning (a form of electroplating), and cementation. Ion exchange and solvent extraction are often used to concentrate values before recovery.

Ores can be exposed to leach in a variety of ways. In vat leaching, the process is carried out in a container (vat), often equipped with facilities for agitation, heating, aeration, and pressurization (e.g., Pachuca tanks). In situ leaching is employed in shattered or broken ore bodies on the surface, or in old underground workings. Leach solution is applied either by plumbing or percolation through overburden. The leach is allowed to seep slowly to the lower levels of the ore body or mine where it is pumped from collection sumps to a metal recovery or precipitation facility. In situ leaching is most economical when the ore body is surrounded by an impervious matrix which minimizes loss of leach solution. However, when water suffices as a leach solution and is plentiful, in situ leaching is economical, even in previous strata.

Low-grade ore, oxidized ore, or tailings can be treated above ground by heap or dump leaching. Dump leaching is usually employed for leaching of low-grade ore. Most leach dumps are deposited on existing topography. The dump site is often selected to take advantage of impermeable surfaces and to utilize the natural slope of ridges and valleys for the collection of pregnant leach solution. Heap leaching is employed to leach ores of higher grade or value. Heap leaching is generally done on a specially prepared impervious surface (asphalt, plastic sheeting, or clay) that is furrowed to form drains and launders (collecting troughs). This configuration is employed to minimize loss of pregnant leach solution. The leach solution is typically applied by spraying, and the launder effluent is treated to recover metal values.

Gold (cyanide leach) and uranium and copper (sulfuric acid leach) are recovered by leaching processes. Leaching is also used in processing ores of ferroalloys, radium, and vanadium.

Amalgamation

Amalgamation is the process by which mercury is alloyed, generally to gold or silver, to produce an amalgam. This process is applicable to free milling of precious metal ores; that is, those in which the gold is free, relatively coarse, and has clean surfaces. Lode or placer gold and silver that is partly or completely filmed with iron oxides, greases, tellurium, or sulfide minerals cannot be effectively amalgamated. Hence, prior to amalgamation, auriferous ore is typically washed and ground to remove any films on the precious metal particles. Although the amalgamation process was used extensively for the extraction of gold and silver from pulverized ores, it has largely been superseded by the cyanidation process due to environmental considerations.

A more complete description of amalgamation practices for the recovery of gold values can be found in Appendix A.

Cyanidation

With occasional exceptions, lode gold and silver ores now are processed by cyanidation. Cyanidation is a process for the extraction of gold and/or silver from finely crushed ores, concentrates, tailings, and low grade mine run rock by means of potassium or sodium cyanide used in dilute, weakly alkaline solutions. The gold is dissolved by the solution according to the reaction:

$$4Au + 8NaCN + 2H_2O + O_2 \text{ --- } 4NaAu(CN)_2 + 4NaOH$$

and subsequently adsorbed onto activated carbon (carbon-in-pulp process) or precipitated with metallic zinc according to the reaction (Reference 1):

$$\begin{gathered}2NaAu(CN)_2 + 4NaCN + 2Zn + 2H_2O \text{ --- } \\ 2Na_2Zn(CN)_4 + 2Au + H_2 + 2NaOH\end{gathered}$$

The gold particles are recovered by filtering, and the filtrate is returned to the leaching operation.

The carbon-in-pulp process was developed to provide economic recovery of gold from low grade ores or slimes. In this process, gold which has been solubilized with cyanide is brought into contact with activated coconut charcoal in a series of tanks. The ore pulp and enriched carbon are air lifted and discharged onto small vibrating screens between tanks, where the carbon is separated and moved to the next adsorption tank. Gold-enriched carbon from the last adsorption tank is leached with hot caustic cyanide solution to desorb the gold. This hot, high grade solution containing the leached gold is then sent to electrolytic cells where the gold and silver are deposited onto stainless steel wool cathodes.

Pretreatment of ores containing only finely divided gold and silver usually includes multistage crushing, fine grinding, and classification of the ore pulp into sand and slime fractions. The sand fraction is then leached in vats with dilute, well-aerated cyanide solution. After the slime fraction has thickened, it is treated by agitation leaching in mechanically or air agitated tanks, and the pregnant solution is separated from the slime residue by thickening and/or filtration. Alternatively, the entire finely ground ore pulp may be leached by agitation leaching and the pregnant solution recovered by thickening and filtration.

When this process is employed, the pulp is also washed by countercurrent decantation (CCD) to maximize the efficiency of gold recovery. A CCD circuit consists of a number of thickeners connected in series. Direction of the overflow through the thickeners is countercurrent to the direction of the underflow (pulp). Wash solution used in the CCD circuit is subsequently dosed with cyanide and used in the agitation leaching process. Pulp moving through the CCD circuit is discharged from the last thickener to tailings disposal.

In all of these leaching processes, gold or silver is recovered from the pregnant leach solutions; however, different types of gold/silver ore require modification of the basic flow scheme. Efficient low-cost dissolution and recovery of the gold and silver are possible only by careful process control of the unit operations involved.

A more complete description of cyanidation practices for the recovery of gold values can be found in Appendix A.

Ion Exchange and Solvent Extraction

These processes are used on pregnant leach solutions to concentrate values and to separate them from impurities. Ion exchange and solvent extraction are based on the same principle: polar organic molecules tend to exchange a mobile ion in their greater charge or a smaller ionic radius. For example, let R be the remainder of a polar molecule (in the case of a solvent) or of a polymer (for a resin), and let X be the mobile ion. Then, the exchange reaction for a uranyltrisulfate complex is:

$$4RX + (UO_2(SO_4)_3 \text{ ---> } R_4\ UO_2\ (SO_4)_3 + 4X$$

This reaction proceeds from left to right in the loading process. Typical resins adsorb about 10 percent of their mass in uranium and increase by about 10 percent in density. In a concentrated solution of the mobile ion (for example, in N-hydrochloric acid), the reaction can be reversed, and the uranium values are eluted (in this example, as hydrouranyl trisulfuric acid). In general, the affinity of cation-exchange resins for a metallic cation increases with increasing valence:

Cr ||| ≠ Mg || > Na|

and, because of decreasing ionic radius, with atomic number:

92U > 42Mo > 23V

The separation of hexavalent 92U cations by ion exchange or solvent extraction should prove to be easier than that of any other naturally occurring element.

Uranium, vanadium, and molybdenum (the latter being a common ore constituent) usually appear in aqueous solutions as oxidized ions (uranyl, vanadyl, or molybdate radicals). Uranium and vanadium are additionally complexed with anionic radicals to form trisulfates or tricarbonates in the leach. Since the complexes react anionically, the affinity of exchange resins and solvents is not simply related to fundamental properties of the heavy metal (U, V, or Mo) as is the case in cationic-exchange reactions. Secondary properties of the pregnant solutions influence the adsorption of heavy metals. For example, seven times more vanadium than uranium is adsorbed on one resin at pH 9; at pH 11, the ratio is reversed with 33 times more uranium than vanadium being captured. Variations in affinity, multiple columns, and leaching time with respect to breakthrough (the time when the interface between loaded and regenerated resin arrives at the end of the column) are used to make an ion exchange process specific for the desired product.

In solvent extraction, the type and concentration of a polar solvent in a nonpolar diluent (e.g., kerosene) affect separation of the desired product. Solvent handling ease permits the construction of multistage, concurrent and countercurrent, solvent extraction concentrators which are useful even when each stage effects only partial separation of a value from an interferent. Unfortunately, the solvents are easily polluted by slime so complete liquid/solid separation is necessary. Ion exchange and solvent extraction circuits can be combined to take advantage of the slime resistance of resin-in-pulp ion exchange and the separatory efficiency of solvent extraction (Eluex process).

Ion exchange and solvent extraction methods are applied in the processing of ores of uranium/radium/vanadium.

ORE MINING METHODS

Metal-ore mining is conducted by a variety of surface, underground, and in situ procedures. The terminology used to describe these procedures has been defined in a United States Bureau of Mines (USBM) mining dictionary (Reference 2).

Surface mining includes quarrying, open-pit, open-cut, open-cast, stripping, placering, and dredging operations. The USBM dictionary definitions provide no clear distinctions between quarrying, open-pit, open-cut, open-cast, and strip mining. A preference can be discerned for using the word "quarrying" in connection with surface mining of stone, although it often is used in connection with surface mining of all construction materials. Strip mining appears to be the preferred term for surface mining by successive parallel cuts that are filled, in turn, with overburden. Red-bed copper in Oklahoma and bauxite in Arkansas are mined in this way.

The terms "open-pit" and "open-cut" identify surface mines other than quarries, strip mines, and placers, but are often applied to these types also. The term open-pit is used more specifically for surface mines in relatively thick ore bodies characterized by permanent disposal of wastes and terraced or benched slopes. Most of the crude ores of copper and iron come from surface mines of this type.

Placer mining, which employs a variety of equipment and techniques including dredging, is used in mining and concentration of alluvial gravels and elevated beach sands. All the illmenite production in Florida and New Jersey is obtained by the dredging of elevated beach sand deposits. Although hydraulic placer mining of gold in California was enjoined by court decree in the 1880's, placering for gold, by other methods, continues in California and Alaska.

Underground mining is conducted through adits or shafts by a variety of methods that include room-and-pillar, block caving, timbered stopes, open stopes, shrinkage stopes, sublevel stopes and others (Reference 3). Underground mining usually is independent of surface mining, but sometimes preceeds or follows it. Waste removal is proportionately much less in underground than in surface mining, but still requires surface waste disposal areas. Underground mines supply substantially all the lead and zinc mined domestically.

In situ mining procedures include the leaching of uranium and copper.

Considerations given to the choice of mining method and brief descriptions of the methods typically employed are given below.

Surface Mining Operations

Whether an ore body will be mined by surface or underground methods will be determined by the economics of the operation. In general, surface or open-pit mining is more economical than underground mining especially when the ore body is large and the depth of overburden is not excessive.

Some predominant advantages inherent in the open-pit method are as follows:

a) The open-pit method is quite flexible in that it often allows for large increases or decreases in production on short notice without rapid deterioration of the workings.

b) The method is relatively safe. Loose material can be seen and removed or avoided. Crews can be readily observed at work by supervisors.

c) Selective mining is usually possible without difficulty. Grade control can be easily accomplished by leaving lean sections temporarily unmined or by mining for waste.

d) The total cost of open-pit mining, per ton recovered, is usually only a fraction of the cost of underground mining. Further, the cost spread between the two methods is growing wider as larger-scale methods are applied to open pits.

Drilling

Drilling is the basic part of the breaking operation in open-pit mining; considerable effort has therefore been expended to develop equipment for drilling holes at the lowest possible cost. There are several types of drills which can be used. These include churn drills, percussion drills, rotary drills and jet-piercing drills. All are designed with one objective in mind: to produce a hole of the required diameter, depth, and direction in rock for later insertion of explosives.

Blasting

Basically, explosives are comprised of chemicals which, when combined, contain all the requirements for complete combustion without external oxygen supply. Early explosives consisted chiefly of nitroglycerine, carbonaceous material and an oxidizing agent. These mixtures were packaged into cartridges for convenience in handling and loading into holes. Many explosives are still manufactured and packaged to the basic formulas.

In recent years, it has been discovered that fertilizer-grade ammonium nitrate mixed with about six percent fuel oil could be detonated by a high explosive primer. This new application has spread to the point where virtually all open-pit mining companies use this mixture (called ANFO) for some or all of the primary blasting.

Typicaly, multiple charges are placed in two rows of drill holes and fired simultaneously to break free the upper part of the face and prevent "back break" beyond the line of holes. In horizontally bedded deposits, the face is often held right on the line of holes. Fifty feet is almost standard for the height of a bench in hard ground, but 30-foot and 40-foot benches are common. The height depends on the reach of the shovel and the character of the ore.

Stripping

Material overlying an ore body may consist of earth, sand, gravel, rock or even water. Removal of this material generally falls under the heading of stripping. Normally, stripping of rock will be considered a mining operation. Generally, stripping will be accomplished with heavy earth-moving equipment such as large shovels, mounted on caterpillar treads and driven by electricity or diesel power, and bulldozers. In the past, railroad cars were used to haul the stripped material to the dump area. Now, however, they have been replaced by large trucks except for situations where long hauls are required.

Loading

After the ore has been broken down, it is transferred to the mill for treatment. In small pits various kinds of small loaders, such as scrapers or tractor loaders, are sometimes used, but in most places the loading is done with a power shovel. Tractors equipped with dozer blades are used for pushing ore over banks so that it can be reached by the shovel, but their most effective use is in cleaning up after the shovel, pushing loose ore back against the toe of the pile where it can be readily picked up on the next cut.

Underground Mining Operations

Historically, the mining method most often used has been some form of open stope. Generally, to reach the ore a shaft is sunk near the ore body. Horizontal passages are cut from the shaft at various depths to the ore. The ore is then removed, hoisted to the surface, crushed, concentrated and refined. Waste rock or classified mill tailings may be returned to the mine as fill for the mined-out areas or may be directed to a disposal basin (tailings area).

Caving systems of mining ore have been developed as economical approaches to mining extensive low-grade ore bodies.

The Shaft

The shaft is the surface opening to the mine which provides a means of entry to or exit from the mine for men and materials, and for the removal of ore or waste from underground to the surface. It may be vertical or inclined. (A passageway or opening driven horizontally into the side of a hill generally for the purpose of exploring or otherwise opening a mineral deposit is called an adit. Strictly speaking, an adit is open to the atmosphere at one end).

With the advent of modern day mining equipment which has greatly increased the speed of shaft sinking it is presently more economical to sink deep hoisting shafts, and vertical shafts are preferred to inclines. In the U.S., mines have ranged to a depth of 2286m (7500 feet). Although it is unusual for a single shaft to be deeper than 1219 meters (4000 feet), one shaft has been sunk to a depth of 2286 meters (7500 feet) in the Coeur d'Alene Mining District of Northern Idaho and another has exceeded 2620 meters (8600 feet) in South Dakota.

In the United States, what are known as "square shafts," which have two skip compartments and one or two large cage compartments, are now the most popular, because they allow the use of large cages, on which mine timbers can be taken into the mine on trucks without rehandling. These shafts have the additional advantage of getting the crew into the mine and out again in a relatively short time. Shafts sunk from underground levels are called winzes. Winzes are established to permit mining at deeper depths.

Shaft conveyances include buckets, skips, cages or skip-cage combinations. The first two are for hoisting rock or ore and they vary in load capacity from one to eighteen tons. They travel at approximately 610-914 meters (2000-3000 feet) per minute. Cages are used for men and materials and can transport as many as 85 men per load at slower speeds. Safety devices exist to prevent shaft conveyances from falling, should cables fail.

There are generally two types of hoists in use. The Koepe or friction-drive hoist, in common use in Europe since 1875, was first introduced to North America approximately two decades ago. Many are now in service. In this type of hoisting operation, ropes (cables) pass over a drum with counter-balancing weights or loads on either side. These are raised or lowered via friction between the ropes and the drum treads on which they rest. The ropes pass over the drum only once. The arc of contact between rope and drum is normally 180 degrees. On the conventional drum-winder hoist the rope is wound onto the drum and, as such, loads are raised or lowered by a simple winding or unwinding operation.

Levels

Levels are horizontal passes in a mine. They are generally driven from the shaft at vertical intervals of 100-200 feet. That part of the level driven from the shaft to the ore body is known as the crosscut, and that part which continues along the ore body is known as a drift. Crosscuts and drifts vary in size from about 2' x 7' to about 9' x 16' depending on the size of the haulage equipment in use. A raise is an opening made in the back (roof) of a level to reach the level above.

Stopes

A stope is an excavation where the ore is drilled, blasted and removed by gravity through chutes to ore cars on the haulage level below. Stopes require timbered openings (manways) to provide access for men and materials. Normally, raises connect a stope to the level above and are used for ventilation, for convenience in getting men and materials into the stope, and for admitting backfill.

Stope Mining Methods

Today more than half the metallic ore produced from underground methods is mined by open stopes with rooms and pillars.

Nearly all of the lead, zinc, gold, and silver mined from underground in the U.S. is mined by this method as well as much of the uranium and some copper and iron. The three commonly used stoping methods are cut-and-fill stoping, square-set stoping, and shrinkage stoping. The stoping method used normally depends on the stability of the walls and roof as ore removal progresses.

The cut-and-fill stope is used in wider irregular ore bodies where the walls require support to minimize dilution (i.e. waste from walls falling into the broken ore). In its simplest form this mining system consists of blasting down a horizontal cut across the vein for a length of 15 meters (50 feet) or more, removing the broken ore and filling the opening thus made with waste (or mill tailings) until it is high enough to attack the back again. Chutes and manways are raised prior to each addition of fill. Waste material (fill) is dumped into the stope through a waste-pass raise to the surface until it is level with the chutes and manways. Flooring (wood or concrete) is placed over the fill before the next ore cut is drilled and blasted. Scrapers or diesel endloaders are used to remove ore to the chutes and to level the waste backfill in the stope.

The square-set stope is used in an ore body where the walls and ore require support during ore removal. After each blast, square-set timbers are erected and made solid by blocking to the walls and back. Square-sets alone will not support large blocks of ground, and therefore their primary function is to serve as a working platform for the miners and as a protection from falling ground. Consequently, square-sets have become recognized as a system of timbering rather than a system of mining. In good practice, not more than two sets high are allowed to stand open. On the top floor the ore is drilled and blasted, and is allowed to fall to the floor below, where it is shoveled into chutes.

The chutes and manways are raised and backfill is placed in the stope as in the cut-and-fill method.

The shrinkage stope is used chiefly in narrow regular ore bodies where the walls and ore require little support. After each blast, sufficient ore is pulled from the chutes to make room for the miners to drill and blast the next section. As the stope progresses upwards the manways are raised slightly above the level of the broken ore. When the stope reaches the level above, it will be full of broken ore. On removal of the ore, the stopes may be filled with waste material.

Undercut Block Caving Mining Method

This system of mining is applicable to large thick deposits of weak ore which are undercut by a gridwork of drifts and crosscuts. The small pillars thus blocked out are reduced in size until they cave, and the whole mass is allowed to settle and crush. A variation of this used in some places relies on induced caving, blasting being used to start the ore movement.

Generally, 91 meters (300 feet) is an economical height for caving and ore is mined in panels in a retreating system. In each panel the ore is undercut on a sublevel, the width of the unsupported section depending on the strength of the ore.

In thick ore an elaborate system of branch raises carries the broken ore to the main level, caving being regulated by the amount of ore drawn off through finger raises immediately under the undercutting level.

In thinner ore bodies, and in places where such an elaborate system of branch raises is not justified, various expedients are used instead of the branch raises. Scrapers in transfer drifts, pulling the ore from finger raises to main chutes, are one successful approach, and shaking conveyors for the same purpose are another.

Undercut caving has been one of the most successful and revolutionary of the new mining systems, and by its reduction in cost has changed tremendous quantities of what would otherwise be waste rock into profitable ore.

Sublevel Caving Mining Method

The sublevel caving method is somewhat similar to block caving except that it is adaptable to smaller more irregular deposits and to softer, stickier ore. The ore is mined downward from a series of sublevels, using fan blasts to break the ore. Since only the sublevels must be kept open, the method is applicable to heavy ground. The capping must cave easily but should not break fine in comparison to the ore or excessive dilution may result.

Placer Mining Operations

Placer deposits consist of alluvial gravels or beach sands containing valuable heavy minerals. In Alaska and parts of the Northwestern U.S., placer deposits are mined for gold (minor amounts of platinum, tin and tungsten may also be recovered). Two basic mining methods are employed. The most widely used method is the use of heavy earth moving equipment such as bulldozers, front-end loaders and backhoes to push or carry the pay gravels to a sluicebox. Generally, either a backhoe is used to load the gravels into the sluicebox or the sluicebox is situated such that the gravel can be pushed directly into its head-end by a bulldozer. The same earth moving equipment is used to strip overburden when required.

At a few sites in Alaska, bucket-line dredges are still used to mine gold from placers. Prior to dredging, the frozen gravels are thawed by circulating water through 3.8 cm (1.5 inches) pipes contained in drill holes spaced on 4.9 meter (16 foot) centers and drilled to bedrock. Thawed gravels are dredged with a chain of buckets which dump their contents into a hopper on the dredge.

Titanium minerals contained in sand deposits in New Jersey and ancient beach placers in Florida are also mined by dredging methods. In these operations, a pond is constructed above the ore body, and a dredge is floated on the pond. The dredges currently used are normally equipped with suction head cutters to mine the mineral sands.

Solution Mining

In situ or solution mining techniques are used in some parts of Arizona, Nevada and New Mexico to recover copper and in Texas to recover uranium. In situ mining involves leaching the desired metal from mineralized ground in place. During in situ leaching, the ore body must be penetrated and permeated by the leaching solution, which must flow through the mineralized zone and then be recovered for processing at the surface. An impermeable underlying bed, such as shale or mudstone, is desirable to prevent downward flow below the ore zone. Usually, in the solution mining of copper, abandoned underground ore bodies previously mined by block caving methods are leached. Although, in at least one case, an ore body on the surface of a mountain was leached after shattering the rock by blasting. In underground workings, leach solution (dilute sulfuric acid or acid ferric sulfate) is delivered by sprays, or other means, to the upper areas of the mine and allowed to seep slowly to the lower levels from which the solution is pumped to a precipitation plant at the surface.

Solution mining of uranium generally involves the leaching of previously unmined, low-grade ore bodies. Injection and production wells are drilled through the mineralized zone, the drilling density depending on the nature of the body. The ore body may be fractured to improve permeability and leachability. The leaching solution is generally a dilute acid or carbonate. An oxidant, such as sodium chlorate, may be added to improve leaching, and a flocculant may improve flow.

INDUSTRY PRACTICE

The processes discussed above are used variously throughout the ore mining and milling category. A profile of the mining and milling practices used in each subcategory follows.

Iron Ore Subcategory

General

American iron ore shipments increased from 1968 to 1973. In 1973, the United States shipped 92,296,400 metric tons (101,738,320 short tons) of iron ore. Shipments declined to a level of 76,897,300 metric tons (84,763,893 short tons) in 1975 and leveled off in 1976 to 77,957,500 metric tons (85,932,552 short tons) (Reference 4). Iron ore shipments decreased to 54,918,000 metric tons (60,539,000 short tons) in 1977. Shipments increased to 84,538,000 metric tons (93,191,000 short tons) in 1978 and to 87,597,000 metric tons (96,564,000 short tons) in 1979 (Reference 5). The general trend in the iron ore industry is to produce increasing amounts of pellets and less "run of mine" quantities (coarse, fines, and sinter). Total pellet production in 1976 was 68,853,800 metric tons (75,897,543 short tons), or 88.3 percent of all iron ore shipped, whereas only 70 percent of all iron ore shipped in 1973 was in the form of pellets.

Based on production figures, 54 percent of the U.S. iron ore industry uses milling operations which result in no discharge, 31 percent discharge to surface waters, and the discharge practices for 15 percent are unknown. For pelletizing operations alone, 56 percent of total production is represented by operations practicing no discharge of process wastewater, 35 percent discharge to surface waters, and the discharge practices of 8 percent are unknown. A summary presented in Tables III-1 and III-2 shows production data, processes and wastewater technology employed, and discharge methods and volumes.

Unlike the milling segment, the mining segment of the iron ore industry does discharge, either directly to the environment or into the mill water circuit, either as the primary source of process water or as makeup water. Water can cause a variety of problems if allowed to collect in mine workings. Therefore, water is collected and pumped out of the mine.

The primary discharge water treatment used in mining and mining/milling operations is removal of suspended solids by settling. A single facility uses alum and a long-chain polymer as flocculation aids for fine-grained suspended solids.

In 1978, one facility (Mill 1113) which formerly discharged was expected to achieve no discharge of process wastewater. This facility accounted for approximately 13 percent of the total U.S. production of iron ore pellets. In 1978, approximately 69 percent of iron ore pellet production will come from zero-discharge facilities, and zero-discharge for the Mesabi Range subcategory is required for BPT.

Recent Trends

A new technology to obtain an acceptable iron ore product has been developed recently and is currently being used at Mill 1120. If successful, it could result in a shift from the current trend of mining magnetic taconite ores to the mining of fine-grained hematite ores. Due to the fine-grained nature of these ores (85 percent is less than 25 micrometers (0.001 inch)), very fine grinding is necessary to liberate the desired mineral. Conventional flotation techniques used for coarse-grained hematite ores have proven unsuccessful because of the slimes developed by the fine-grinding process.

A selective flocculation technique has been developed that reduces the slimes which are so detrimental. In this process, the iron minerals are flocculated selectively from a starch product while the siliceous slimes are dispersed using sodium silicate at the proper pH. After desliming, cationic flotation is used, incorporating an amine for final upgrading. A simplified flow sequence for liberation of the fine-grained hematites is illustrated in Figure A-4 of Appendix A. Careful control of water hardness is necessary for the process to function properly. Recycled water is lime-treated to create a water-softening reaction.

Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum

This subcategory includes many types of ore metals which are milled by similar processes and which have similar wastewaters.

Copper Mining

Based on the profile of copper mines shown in Table III-3, there are presently 33 operations engaged in the mining of copper. This listing includes those operations whose status is active, exploratory, or under development. The vast majority of the mines listed in the tables are located in Arizona, while the others are located in seven other states. In addition to the mines listed in Table III-3, a recent MSHA (Mine Safety and Health Administration) tabulation indicates that 22 smaller operations with an average employment of about 10 people are presently engaged in copper mining. However, production, process and water use information for these small mines is not available.

The tabulation below provides a production cross section of the major copper mining states in 1976 (Reference 6):

State	Production	
	1000 Metric Tons	1000 Short Tons
Arizona	157,339	173,472
Utah	26,817	29,567
New Mexico	22,690	25,016
Montana	15,220	16,781
Nevada	7,092	7,820
Michigan	3,448	3,801
Tennessee	1,845	2,034
Idaho	128	141

The total domestic copper mine production from 1968 to 1979 is shown below (1968-1973 production - Reference 7, 1974-1976 production - Reference 6, 1977-1979 production - Reference 5):

Year	Production	
	1000 Metric Tons	1000 Short Tons
1968	154,239	170,054
1969	202,943	223,752
1970	233,760	257,729
1971	220,089	242,656
1972	242,016	266,831
1973	263,088	290,000
1974	266,153	293,443
1975	238,544	263,003
1976	257,349	283,736
1977	235,844	259,973
1978	239,247	263,724
1979	264,790	291,881

As shown in Table III-3, 19 operations employ surface mining methods, while 10 operations mine underground. Four operations use both methods. The U.S. Bureau of Mines reports that 84 percent of the copper and 90 percent of the copper ore was produced from open-pit mines in 1976.

Water handling practices at most mines result in the use of mine water as makeup for leaching or milling circuits. However, mine water is discharged to surface waters (direct discharge) at as a result of "dormant" operational status. Mine water discharge practices at seven operations are unknown because of exploratory or development status.

Many western copper mines use leaching operations. Leaching operations currently employ sulfuric acid (5 to 10 percent) or iron sulfate to dissolve copper from the oxide or mixed oxide-sulfide ores in dumps, heaps, vats or in situ. The copper is subsequently recovered from solution in a highly pure form via precipitation, electrolytic deposition (electrowinning), or solvent extraction-electrowinning. Production of cement and electrowon copper contributes a significant quantity (17.6 percent in 1976) of the recoverable copper produced through mining.

Since leaching circuits require makeup water, total recycle of leach circuit water is common practice. Therefore, the BPT effluent limitations guidelines for mines and mills which employ dump, heap, in situ, or vat leach processes for the extraction of copper from ores or ore waste was no discharge of process wastewater. A clarification of the limitations was also promulgated (44 FR 7953) which addressed the intent of the limitation, the applicability of relief from effluent limitations, and defined areas of coverage.

Copper Milling

Nearly all copper mines are associated with mills, as seen in the copper mill profile (Table III-4). Froth flotation, a process designed for the extraction of copper minerals from sulfide ores, is the predominant ore beneficiation technique. The ore is crushed and ground to a suitable mesh size and is sent through flotation cells. Copper sulfide concentrate is lifted in the froth from the crushed material and collected, thickened and filtered. The final concentrate from the mill may contain 15 to 30 percent copper.

Many metal byproducts are claimed from the copper concentrates produced at mills; however, most byproduct values are realized at smelters and refineries. A major byproduct associated with the copper mills is molybdenum concentrate, and molybdenum containing byproducts were reported from 14 of the mills in 1971.

The final concentrate from the mill is sent to the smelter for production of blister copper (98 percent Cu). The refinery produces pure copper (99.88 to 99.9 percent Cu) from the blister copper, which retains impurities such as gold, silver, antimony, lead, arsenic, molybdenum, selenium, tellurium, and iron. These impurities are removed at the refinery.

Milling wastewater handling practices differ throughout the industry, but most operations recycle mill water due to a negative water balance (net evaporation). Only two milling operations, both in the East, recycle a minor portion or none of the milling wastewater. Four ore beneficiation operations practice discharge to surface waters. One operation combines mine, leach and mill wastewater and reports a discharge. At least one milling operation is reported to discharge intermittently.

Lead/Zinc

Lead and zinc are often found in the same ore and as such, are discussed together throughout this document. The domestic lead/zinc mining industry presently consists of 29 mining operations (which may have more than one mine) and 23 concentrators located in 12 states. Three of these mines and four of the concentrators have begun production since 1974. This increased production capability has been offset somewhat by the closing of seven mines and six concentrators during the same time period.

U.S. mine production of lead in 1975 dropped six percent to 565,000 metric tons (621,500 short tons) from a record high of 603,500 metric tons (663,900 short tons) achieved in 1974 (Reference 6). Lead production has continued to decline. Production in 1977 totaled 537,499 metric tons (592,500 short tons) (Reference 5). Production of lead continued to decline and in 1979 production totaled 525,569 metric tons (579,300 short tons) (Reference 5). Missouri remained the leading producer, with 89.8 percent of the nation's total lead production, followed by Idaho, Colorado, and the other states. The seven leading mines, all in Missouri, contributed 79 percent of the total U.S. mine production, and the 12 leading mines produced 91 percent of the total (Reference 8). Although lead production declined between 1974 and 1979, domestic lead prices continued to rise from a price of 47.4 cents per kilogram (21.5 cents per pound) in 1975 to $1.16 per kilogram (52.6 cents per pound) in 1979. The current price (15 September 1981) is 93 cents per kilogram (42 cents per pound) (Reference 9).

Domestic mine production of zinc decreased from 454,500 metric tons (499,900 short tons) in 1974 to 426,700 metric tons (469,400 short tons) in 1975. It then recovered to about 440,000 metric tons (484,500 short tons) in 1976 (Reference 5). Zinc production decreased to 407,900 metric tons (449,600 short tons) in 1977 and further to 302,700 metric tons (333,700 short tons) in 1978 and 267,300 metric tons (294,600 short tons) in 1979 (Reference 5). Tennessee was the leading zinc producing state in 1979 with 32 percent of total production and was followed by Missouri, 23 percent; New Jersey, 12 percent; and Idaho, 11 percent (Reference 5). Tennessee led the nation in production of zinc for 15 consecutive years prior to 1973 and regained that status in 1975, 1977, 1978, and 1979 due to the opening of two new concentrations.

In contrast to climbing lead prices, zinc prices have followed a downward trend over the past decade in terms of real dollars. Following inflationary price increases during the period 1972 to 1975, zinc prices declined from the average 1975 price of 85.91 cents per kilogram (38.96 cents per pound) to 81.61 cents per kilogram (37.01 cents per pound) in 1976 to 75.85 cents per kilogram (34.4 cent per pound) in 1977 and then increased to the current (15 September 1981) price of $1.09 per kilogram (49.25 cents per pound) (Reference 9).

The domestic mine production of lead/zinc continues to come chiefly from ores mined primarily for their lead and zinc content. Less than 1 percent of the total lead/zinc production is derived as byproduct or coproduct of ores mined for copper, gold, silver, or fluorspar (Reference 10). The complex ores of the Rocky Mountain area are particularly dependent on economic extraction of the mineral's aggregate metal value, and the metal of highest value is variable. Byproduct recovery at the smelter or refinery from the processing of lead and zinc concentrates also yields a significant portion of the domestic production of antimony, bismuth, tellurium, and cadmium.

Lead and zinc ores are produced almost exclusively from underground mines. Several mines began as open-pit operations and have developed into underground mines. Conversely, a number of underground mines have surfaced while following an ore body, resulting in small, open-pit operations. At present, only one open-pit mine is in operation, and it is actually the intersection of an underground lead/zinc ore body with an adjacent open-pit copper mine.

A general description of the lead/zinc mining industry is contained in Tables III-5 to III-8 and includes processes, products, location, age, wastewater treatment technology, and discharge method and volume (see also References 11 and 12). As previously indicated, many of the mines and mills shown as lead/zinc also mine or mill for other metal values that are interspersed within the lead/zinc ore matrix. These metal values are usually copper, gold, or silver. However, the mines and mills shown as lead/zinc are characterized based on their primary products, lead and zinc.

Gold

The four leading U.S. gold producers accounted for 73 percent of total production during 1975. Approximately 95 percent of all production came from 25 mines or mine/mill operations, 10 of which were operated primarily for recovery of gold (Reference 8). Thirty-six percent of total production was recovered as a byproduct of other mining (for example, where copper or lead/zinc are primary metals); the remainder was recovered at gold lode and placer operations.

Gold prices have risen significantly in the past year. The price of gold on the open market reached a previous high of nearly $200 per 31.1 gram (1 troy ounce) during 1974; in January 1980 the price was over $800.00 (Reference 13) per 31.1 gram (1 troy ounce). It is currently (15 March 1982) $315 per 31.1 gram (1 troy ounce) (Reference 9). An expected response to the high price of gold is the increased gold prospecting and development activities reported in gold mining areas of the country (Reference 14). In addition, plans for significant investment in new production facilities or renovation of inactive facilities have been announced by a number of mining companies during the past 3 to 4 years. During 1975 and 1976, four to six new cyanidation operations began full scale operation. However, despite these prospects, domestic production has continued to decline during recent years. Reported domestic production for 1975 was 32.7 metric tons (1,052,000 troy ounces), a decline of 27 percent from reported production of 45.1 metric tons (1,450,000 troy ounces) in 1972.

The steady decline in domestic production of gold is due to several factors: (1) inflation and shortages of equipment, material, and labor have limited new mine developments; (2) in most instances, lower grade ores are being mined, but mine and mill limitations have generally allowed little expansion of tonnage handled; (3) diminished copper production due to low copper prices in 1977 to 1978 led to a decline in byproduct production of gold; and (4) depletion of ore at two major producing gold lode operations has resulted in the suspension of all production at one during October 1977, while the second is scheduled for permanent closure. These two mine/mill operations accounted for 18 percent of reported domestic primary gold production in 1974.

A summary description of gold mine/mill operations is presented in Tables III-9 to III-12. As indicated, most operations employ the cyanidation process for recovery of gold (see description under Ore Beneficiation Processes) and Appendix A for specific applications). This is especially true of lode mining operations which have recently become active and are located predominantly in Nevada. At these sites, heap leaching or agitation leaching processes have been the methods of choice. In addition, the preferred process for recovery of the gold from solution has been the recently developed carbon-in-pulp process (see Appendix A). The simplicity of its operation and the low capital and operating costs have made this process economically superior to the conventional zinc-precipitation process. This factor and the current high selling price of gold have served to make development and mining of some small or low-grade gold ore bodies economically feasible.

Spent leach solutions are recycled at cyanidation leaching operations. This practice has generally been implemented for conservation of both reagent and process water.

Of the lode mill operations operating for the primary recovery of gold, two report discharge of wastewater. One of these is building facilities which will provide the equivalent of zero discharge of mill wastewater. This mill uses the cyanidation process and under the BPT regulation, zero discharge of process wastewater is required.

The second lode mill operation which discharges wastewater recovers gold by amalgamation and lead/zinc by flotation. The alkaline wastewater which results is settled in a multiple pond system prior to final discharge.

The exact number of lode gold mines which have a discharge and are not directly associated with a mill is not known. Treatment of wastewater at placer mining operations is often not practiced. At the large dredging operations and at the smaller hydraulic and mechanical excavation operations, settling ponds have been provided.

Silver

Domestic mine production of silver in 1975 totaled 1,085.4 metric tons (34.9 million troy ounces). The largest percentage of this production continued to be a byproduct of base-metal mineral mining. During 1974, only three of the 25 leading producers mined ore primarily for its silver content. These three were the first, second, and eighth leading producers and accounted for approximately 30 percent of total domestic primary production. Production at a fourth major silver mine was curtailed during 1973 due to depletion of known ore reserves. Two recently developed silver mine/mill operations began operation during 1976 and are expected to become major producers.

The selling price of silver, like that of gold, reached an all-time high during 1979. During January 1980, the selling price reached $48.00 (Reference 13) per 31.1 grams (1 troy ounce). The current price of silver (15 March 1982) is $7.20 per 31.1 grams (1 troy ounce) (Reference 9).

A summary description of silver mine/mill operations is presented in Tables III-13 and III-14. More than 300 U.S. mines supply ore from which silver is recovered. However, as previously stated, most of this ore is exploited primarily for its copper, lead, or zinc content. Byproduct silver is typically recovered from base-metal flotation concentrates during smelting and refining processes. The large operations which are exploiting ore primarily for its silver content also recover the sulfide minerals by flotation. Only a small fraction of a percent of total silver production is recovered from placer mining or by amalgamation. Approximately 1 percent is recovered by the cyanidation process.

Wastewater treatment practices at major silver mine/mill operations typically employ a pond for collection and retention of the bulk flotation-circuit tailings. In some instances, multiple pond systems are employed to optimize control of suspended solids. At one of the four flotation mills operating during 1977, clarified decant was recycled from the tailing pond to the mill. Partial recycle is practiced at a second operation. Two mills situated in a river valley in Idaho presently discharge to a common tailing pond system.

A major silver producing operation which opened in 1976 is employing a vat leach (cyanidation), zinc-precipitation circuit. Wastewater generated in this process is impounded and recycled for reuse within the mill.

At one operation, mine drainage is settled in a multiple pond system prior to final discharge and at a second operation, minewater is directly discharged without treatment. Minewater at all other silver mining operations (for which information is available) is either used as makeup in the mill or is discharged into the mill tailing pond treatment system.

Platinum

The platinum-group mining and milling industry includes those operations which are involved in the mining and/or milling of ore for the primary or byproduct recovery of platinum, palladium, iridium, osmium, rhodium, and ruthenium. Domestic production of these platinum-group metals results as a byproduct of copper refining. Until recently, production from a single placer operation in Alaska accounted for all the U.S. production from mining primary ores. In 1976, this facility, located in the Goodnews Bay District of Alaska, ceased operation. Total "mine" production of platinum-group metals in 1978 was 258.2 kilograms (8,303 troy ounces), which was recovered entirely as a byproduct of copper refining. Of this, approximately 39.1 kilograms (1,258 troy ounces) represented platinum metal itself. Total secondary recovery of platinum-group metals (from scrap) was 7,998.6 kilograms (257,191 troy ounces) in 1978.

Platinum-group metals are recovered as secondary metals in many places within the United States. Minor amounts have been recovered from gold placers in California, Oregon, Washington, Montana, Idaho, and Alaska, but significant amounts as primary deposits have been produced only from the Goodnews Bay area in Alaska.

In 1976, the single platinum mining operation employed techniques similar to those used for recovery of gold from placer deposits, i.e., bucketline dredging. The coarse, gravelly ore was screened, jigged and tabled. Chromite and the magnetite were removed by magnetic separation techniques. After drying, air separation techniques were applied, and a 90 percent concentrate was obtained. Water used for the initial processing was discharged from the dredge into a settling pond and subsequently discharged from the pond after passing through coarse tailings. Removal of total suspended solids to below 30 mg/l was accomplished.

In the United States, the major part of platinum production has always been recovered as a byproduct of copper refining in Maryland, New Jersey, Texas, Utah and Washington. Byproduct platinum-group metals are sometimes refined by electrolysis and chemical means to yield recoveries of over 99 percent. The price of platinum as quoted recently (15 September 1981) on the spot market was $475 per 31.1 grams (1 troy ounce) (Reference 9).

Molybdenum

Production of molybdenum has been, generally, increasing over the past 30 years as illustrated below (References 5, 7, 15, 16, 17):

Year	Production Metric Tons	Production Short Tons
1949	10,222	11,265
1953	25,973	28,622
1958	18,634	20,535
1962	23,250	25,622
1968	42,423	46,750
1972	46,368	51,098
1974	51,000	56,000
1977	55,484	61,204
1979	65,302	71,984

Since 1974 significant exploration and development has taken place, and production is expected to increase at a higher rate. Production figures are not available yet, but several new operations have begun since 1976, and a number of mines and mills are in the planning and development stages.

As shown in Table III-15, there are six mines involved with molybdenum; three mines are producing now, and three have exploration underway. The three producing mines are: one open-pit in New Mexico, an underground and a combined open-pit/underground in Colorado. One Colorado operation recovers secondary products of tungsten and tin.

The two Colorado mines discharge to surface waters; one by way of a mill water treatment system, the other by way of separate treatment. The New Mexico mine has no discharge because groundwater is not encountered.

All three active mines are associated with flotation mills which are described in Table III-16. The New Mexico facility treats mill water for discharge by primary and secondary settling (two tailings and one settling pond), and aeration for cyanide removal. They are currently experimenting with hydrogen peroxide for cyanide oxidation. The underground/open-pit Colorado operation uses a complex treatment system including settling, recycle, ion exchange, electrocoagulation flotation, alkaline chlorination and mixed-media filtration. The second Colorado mill accomplishes no discharge by total recycle.

Aluminum

In 1979, the major aluminum ore, bauxite, was mined by eight companies located in Arkansas, Alabama, and Georgia (Reference 5). Arkansas accounted for 79 percent of the total bauxite mined in 1979 (Reference 5). The only operations mining bauxite for aluminum production are two operations located in Arkansas. In both BPT and BAT effluent guidelines, the aluminum ore subcategory applies only to the mining of bauxite for eventual metallurgical production of aluminum. Most bauxite mined at the two Arkansas operations is refined to alumina (Al_2O_3) by the "Combination Process," which is classified as SIC 2819. A gallium byproduct recovery operation is used at one Arkansas operation. Domestic production of bauxite is listed below from 1974 to 1979 (References 5 and 18):

Year	Production Metric Tons	Production Short Tons
1974	1,980	2,181
1975	1,800	1,983
1976	1,989	2,191
1977	2,013	2,217
1978	1,669	1,839
1979	1,821	2,006

Average annual production for the last 10 years is approximately 1,880,000 metric tons (2,070,000 short tons).

All production from the two domestic aluminum ore operations originated from open pits. The sole underground bauxite mine closed in late 1976. Bauxite ore used for refining alumina is graded on silica content, and the percentage of domestic bauxite shipments by silica content is listed below (Reference 5):

Silica (SiO_2) Content (%) of Ore	Percentage of Total Domestic Shipments 1975	1976	1977	1978	1979
less than 8	4	6	2	2	1
8 to 15	62	50	54	55	55
greater than 15	34	44	44	43	44

A pilot project proved the economic viability of alunite as an alternate ore for production of alumina, but construction of a commercial scale refinery in Utah has not begun. The mine and refinery complex was expected to produce alumina, potassium sulfate, and sulfuric acid when completed (Reference 19).

Both domestic bauxite ore operations require discharge of large volumes of mine water, and there is no process water used for crushing or grinding of the ore. The total daily discharge of mine water attributable to bauxite ore mining is about 40,000 cubic meters (10.6 million gallons), with about 80 percent of the discharge flow attributed to a single operation. Characteristics of the two domestic bauxite operations are shown in Table III-17.

Tungsten

Tungsten mining and milling is conducted by numerous (probably more than 50) facilities, the majority of which are very small and operate intermittently. As illustrated below, tungsten production was relatively constant between 1969 and 1979 (References 5, 18, and 20):

	Domestic Production (Contained Tungsten)	
Year	Metric Tons	Short Tons
1969	3,543	3,903
1970	4,369	4,813
1971	3,132	3,450
1972	3,699	4,075
1973	3,438	3,787
1974	3,350	3,690
1975	2,536	2,794
1976	2,646	2,915
1977	2,727	3,004
1978	3,130	3,448
1979	3,015	3,321

A profile of tungsten mining is presented in Tables III-18 and III-19. Table III-18 describes the larger operations (annual production greater than 5,000 metric tons ore processed/year), and Table III-19 describes smaller operations (production less than 5,000 metric tons ore processed/year). The majority of the mines are located in the western states of California, Oregon, Idaho, Utah, and Nevada. Almost all are underground mines, and many have no discharge of mine water.

Tables III-20 and III-21 profile tungsten mills. Processes used are gravity separation and/or fatty acid and sulfide flotation. One mill in California produces the majority of the tungsten concentrate. Wastewater treatment methods vary, but may include settling (tailing ponds) and recycle and/or evaporation. Most of the active mills do not discharge, primarily because they are in arid regions and need the water.

Mercury

During recent years, the domestic mercury industry has been characterized by a general downward trend in the number of actively producing mines or mine/mill operations. Historically, mine output in the United States has come from a relatively large number of small production operations. However, since 1969, the number of actively producing mines has declined from 109 to just 4 in 1980 (Reference 19).

Domestic primary production during 1980 was 30,657 flasks (34.5 kilograms (76 pounds) per flask) (Reference 21). More than 75 percent was produced by a single mine/mill in Nevada which began operation during 1975. Byproduct recovery was reported at a gold mining operation in Nevada. The total domestic production of mercury during 1980 came from two mines in California and two mines in Nevada.

The primary factor contributing to the recent depression of the domestic mercury industry was a steady decline in the selling price of mercury during the late 1960's and early 1970's. Between 1968 and 1975, the selling price decreased to an average of $117 per flask in New York. However, as of February 1982, the price had risen to about $400 per flask. Additional factors having an adverse impact include: (1) widely fluctuating prices caused by erratic demand, (2) competition from low-cost foreign producers, and (3) the low grade ore resulting in high production costs. A descriptive summary of active mercury mine and mill operations is presented in Tables III-22 and III-23.

The majority of U.S.-produced mercury is recovered by a flotation process at one mill in Nevada. Ore processed in that mill is mined from a nearby open pit. The flotation concentrate produced is furnaced on site to recover elemental mercury. Wastewater treatment consists of impoundment in a multiple pond system with no resulting discharge. The majority of impounded wastewater evaporates, although a small volume of clarified decant is occasionally recycled.

A second operation, located in California, employs a gravity concentration process. Ore is obtained from an open-pit mine and the concentrate is furnaced on site to produce elemental mercury. Wastewater is settled and recycled during the 9 months of the year that the mill is active. During the remaining 3 months (winter months), however, the mill is inactive, and a mine water discharge from the settling pond often occurs as a result of rainfall and runoff. This facility is presently inactive.

An additional number of small operations, located in California and Nevada, operate intermittently. Ore is generally furnaced directly without prior beneficiation. Water is not used except for cooling in the furnace process.

Uranium

This category includes facilities which mine primarily for the recovery of uranium, but vanadium and radium are frequently found in the same ore body. Uranium is mined chiefly for use in generating energy and isotopes in nuclear reactors. Where vanadium does not occur in conjunction with uranium/radium (nonradioactive), it is considered a ferroalloy and is discussed as a separate subcategory. Within the past 20 years, the demand for radium (a decay product of uranium) has vanished due to the availability of radioactive isotopes with specific characteristics. As a result, radium is now treated by the industry as a pollutant rather than as a product.

Primary deposits of uranium ores are widely distributed in granites and pegmatites. These black ores contain the tetravalent minerals uraninite (UO_2) and coffinite ($U(SiO_4)_{1-x}(OH)_{4x}$) with pyrite as a common gangue mineral. Secondary, tertiary, and higher order uranium deposits are found in relatively shallow sandstones, mudstones, and limestones. These deposits are formed by the transport of soluble hexavalent uranyl compounds (notably carbonates) with the composition U_3O_8 (i.e., $UO_2.2UO_3$) being particularly stable. Transport of the uranyl compounds leads to the surface uranium ores commonly found in arid regions, including carnotite ($K_2(UO_2)_2(VO_4)_2.13H_2O$), uranophane ($Ca(UO_2)(SiO_3)_2(OH)_2.\ 5H_2O$), and autunite ($Ca(UO_2)_2(PO_4)_2.10\text{-}12H_2O$). If reducing conditions are encountered, as in subsurface sedimentary deposits, tetravalent uranium compounds are redeposited.

The major deposits of high-grade uranium ores in the United States are located in the Colorado Plateau, the Wyoming Basins, and the Gulf Coast Plain of Texas. In 1976, New Mexico provided 46 percent; Wyoming, 32 percent; and Utah, Colorado, and Texas, the remaining 22 percent of total U.S. production (Reference 22). Total domestic production of uranium for 1977 was predicted to be almost 9,100,000 metric tons (10,000,000 short tons) of ore, with an average grade of 0.15 percent U_3O_8 (Reference 22). Average ore grade is down from 0.17 percent in 1975 and 0.18 percent in 1974, reflected in increases in the price of U_3O_8 from $77 per kg ($35 per pound) in 1975 to $92 per kg ($42 per pound) in 1977 (References 23 and 24).

Tables III-24, III-25 and III-26 present profiles of the uranium mining and milling industry in the United States. The information presented in Table III-24 represents over 90 percent of the total U.S. production of uranium ore. Depending on the nature of the operation, each listing may represent anywhere from one to 40 individual mines. Table III-25 presents information for all active uranium mills in the country. Table III-26 presents available information for in situ uranium mines.

Following is a brief description of the uranium industry. A more detailed account of the processes, water use, wastewater generation, and treatment in the industry may be found in Reference 24.

Mining

Mining practice in the uranium industry is by open-pit or underground. There were approximately 160 underground mines in the United States as of September 1977 (Reference 11), and more than 50 percent have fewer than five employees. There are 53 surface mines in the United States, and about 26 percent of these employ fewer than five people. The actual number of active mines at any given time will vary, depending on market conditions and company status.

Most mines ship ore to the mill by truck. The economics of hauling unbeneficiated ore require that the distance from the mine to the mill be no more than a few kilometers (approximately one mile). However, certain high-grade ores (0.6 percent U_3O_8) may be shipped up to 200 kilometers (120 miles). The large number of small mines often requires individual mills to purchase ore from several different mines, both company and privately owned. A single mill may be fed by as many as 40 different mines.

Milling

As of February 1979, there were 20 active uranium mills in the United States, ranging in ore processing capacity from 450 metric tons per day (500 short tons per day) to 6,300 metric tons per day (7,000 short tons per day). In addition, four of these mills are practicing vanadium byproduct recovery, one mill is recovering molybdenum concentrate as byproduct, and another intermittently recovers copper concentrate. One mill, which historically produced vanadium from uranium ore, is currently producing several vanadium products from vanadium concentrate shipped from a nearby mill. A complete discussion of the milling and extraction technology used in this subcategory is presented in Appendix A.

Byproduct vanadium recovery is practiced at three uranium mills. At Mill 9401, an alkaline mill, purification of crude yellowcake by roasting with soda ash (sodium carbonate) and leaching the calcine with water generates a vanadic acid solution. This solution, which contains about eight percent V_2O_5, is stockpiled and sold for vanadium recovery elsewhere. Mill 9403, which operates an acid-leach circuit, recovers vanadium as a solvent-exchange raffinate. Vanadium values in the raffinate are concentrated and recovered by solvent exchange, precipitation of ammonium vandates (from the pregnant stripping agent) with ammonium chloride, filtration, drying, and packaging. Mill 9405, which also operates an acid-leach circuit, recovers vanadium from ion exchange circuit raffinate. The raffinate is treated with sodium chlorate, soda ash, and ammonia to precipitate impurities and is then directed to a solvent-extraction circuit. Pregnant stripping solution from the solvent extraction circuit, containing the vanadium values, is collected and shipped to a nearby facility for further processing.

In addition to uranium and vanadium, Mill 9403 intermittently recovers copper concentrate from uranium ore high in copper values. Copper recovery includes a sulfuric acid leach, which generates a pregnant liquor containing dissolved uranium as well as copper. The dissolved uranium is recovered in a solvent exchange circuit and then directed to the main plant for final processing and recovery as yellowcake.

In Situ Recovery

Eight operations in Southern Texas are practicing in situ leaching of uranium, and seven more in situ leaching operations are under development. Annual production from six of the eight on-line facilities is estimated to be 671 metric tons (740 short tons) of U_3O_8 (Reference 25). Typically, alkaline leach solutions are pumped into a series of strategically placed injection wells, recovered from a production well, and either shipped to a nearby mill or recovered on site. The uranium is concentrated using fixed bed ion exchange and conventional yellowcake precipitation techniques.

Industry Trends

Although many uncertainties exist about the future use of nuclear energy in this country, increases in yellowcake requirements are expected within the next 10 years. The annual demand for yellowcake is expected to grow from approximately 15,400 metric tons (17,000 short tons) of U_3O_8 in 1976 to 35,400 metric tons (39,000 short tons) in 1985. As a result, the decreasing grade of mined ore will require U.S. mill capacity to increase from 8,890,000 metric tons (9,800,000 short tons) of ore per year to 46,200,000 metric tons (50,800,000 short tons) of ore per year (Reference 21). Because of recent developments, however, the Nuclear Regulatory Commission estimates in 1979 indicated that demand for mill capacity may be in the range of 22 to 27 million metric tons in 1987 and approximately 36.5 million metric tons in the year 2000 (Reference 26).

Projected increases in U_3O_8 demand have resulted in:

(1) the exploration and expansion of known sandstone deposits;

(2) the exploration of new sandstone areas in Nevada, North and South Dakota, Colorado, Wyoming, and Montana; and (3) preliminary investigation of "hardrock" areas in Colorado, Michigan, Wisconsin, Minnesota, the eastern and southwestern United States, and Alaska.

Near-term industry growth patterns include three new uranium mills, rated at 317,000 metric tons (350,000 short tons) per year, 634,000 metric tons (700,000 short tons) per year, and 680,000 metric tons (750,000 short tons) of ore per year, and seven new in situ leach operations.

Increases in the demand and price of yellowcake will provide added impetus for the extraction of lower grade ores, the refinement of conventional milling processes, the development of new mining and milling techniques (e.g., hydraulic mining of sandstones and new milling processes for hardrock ores), and the development and expansion of nonconventional uranium sources such as in situ leaching and phosphate byproduct recovery. Thus, within a few short years, the nature of the uranium mining and milling industry in this country may significantly change.

Water Use and Wastewater Generation

Uranium ores are often found in arid climates, thus water is conserved in milling uranium. Approximately 50 percent of the total U.S. production of uranium ore is recovered from mines which generate mine water. Mine water generation varies from 1.5 cubic meters (390 gallons) per day to 19,000 cubic meters (5,000,000 gallons) per day. Some mines yield an adequate water supply for the associated mill. Those mines which are too far from the mill or which produce water in excess of mill requirements usually treat the mine water to remove pollutants and/or uranium values. Sometimes the treated water is reintroduced into the mine for in situ leaching of values.

The quantity of water used in milling is approximately equal in weight to that of the ore processed. Mills obtain process water from nearby mines, wells and streams. The quantity of makeup water required depends on the amount of recycle practiced, and on evaporation and seepage losses. Eight of the 14 acid mills and three of the four alkaline mills employ at least partial recycle of mill tailing water. The remaining mills employ impoundment and solar evaporation. Acid and alkaline mills, i.e., acid leach, alkaline leach, are explained in detail in Appendix A.

Mine water treatment practices in the uranium industry include: (1) impoundment and solar evaporation, i.e., evaporation by exposure to the sun, (2) uranium recovery by ion exchange, (3) flocculation and settling for heavy metal and suspended solids removal, (4) $BaCl_2$ coprecipitation of radium 226, and (5) radium 226 removal by ion exchange. Discharge is usually to surface waters, which frequently have variable flows depending on seasonal weather conditions.

All uranium mills in the United States impound tailings, which are the primary source of process wastewater in large ponds. Evaporation, seepage, and/or recycle from these ponds eliminate all discharges. One acid mill, however, collects seepage from its tailing pond and overflow from yellowcake precipitation thickeners. This mill then treats the combined waste streams (approximately 2,200 cubic meters (580,000 gallons) per day) to remove radium 226 and total suspended solids (TSS) and discharges to a nearby stream. This facility represents the only known discharging uranium mill in the country.

Antimony

Antimony is recovered from antimony ore (stibnite) and as a byproduct of silver and lead concentrates. This industry is concentrated in two states: Idaho and Montana. Currently, only one operation (Mine/Mill 9901) recovers only antimony ore. The ore is mined underground and concentrates are obtained by the froth flotation process. There is no discharge from the mine, but wastewater from the mill flows to an impoundment. No discharge of process wastewater to surface waters occurs.

A second facility, Mine/Mill 4403 recovers antimony as a byproduct from tetrahedrite, a complex silver-copper-antimony sulfide mineral. The antimony is recovered from tetrahedrite concentrates in an electrolytic extraction plant operated by one of the silver mining companies in the Coeur d'Alene district of Idaho.

Antimony is also contained in lead concentrates and is recovered as a byproduct at lead smelters usually as antimonial lead. This source may represent about 30 to 50 percent of domestic production in recent years.

In 1979, total U.S. mine production of antimony was 655 metric tons (722 short tons). Production at facility 9901 in 1979 was 271 metric tons (299 short tons) of antimony, while production at mine/mill 4403 was 384 metric tons (423 short tons) (Reference 5). In 1979, the total domestic mine production of antimony concentrate was reported as 2,990 metric tons (3,294 short tons). This concentrate contained 655 metric tons (722 short tons) of antimony. Mine/mill 9901 is profiled in Table III-27.

Titanium

The principal mineral sources of titanium are ilmenite ($FeTiO_3$) and rutile (TiO_2). Rutile associated with ilmenite in domestic sand deposits is not separately concentrated typically. The majority of all ilmenite concentrates (includes a mixed product containing ilmenite, rutile, leucoxine and altered ilmenite) produced domestically are from titanium dredging operations. The remainder of the domestic production comes from a mine in New York mining an ilmenite ore.

During recent years, domestic production of ilmenite concentrates has substantially declined. U.S. production of ilmenite during 1968 was 887,508 metric tons (978,509 short tons), while five years later in 1973 production had dropped to 703,844 metric tons (776,013 short tons). Domestic production had dropped to 534,904 metric tons (589,751 short tons) in 1978. The production of ilmenite in the U.S. has declined approximately 40 percent (39.73 percent) between 1968 and 1978. The price of domestically produced ilmenite during early 1973 was approximately $22.64 per metric ton (approximately $23 per long ton) and rose to $54.13 per metric ton ($55 per long ton) by July 1974. The selling price of domestically produced ilmenite has essentially remained at $54.13 per metric ton ($55 per long ton) since 1974, to the present (early 1980). The selling price of domestically produced ilmenite is not significant since the U.S. titanium industry is nearly fully integrated, and most ilmenite concentrates are consumed captively.

A summary description of titanium mine/mill operations is presented in Table III-28 and III-29. As indicated, three of the four active operations employ floating dredges to mine beach-sand placer deposits of ilmenite located in New Jersey and Florida. At these operations, concentration of the heavy titanium minerals is accomplished by wet gravity and dry electrostatic and magnetic methods (see Reference 1 for detailed process description). At the remaining operation, located in New York, ilmenite is mined from a hardrock, lode deposit by open-pit methods. A flotation process is employed in the mill to concentrate the ore materials.

Wastewater treatment practices employed at titanium mine/mill operations are designed primarily for removal of suspended solids and adjustment of pH. In addition, peculiar to the beach sand dredging operations in Florida is the presence of silts and organic substances (humic acids, tannic acids, etc.) in these placer deposits. During dredging operations, this colloidal material becomes suspended, giving the water a deep "tea" color. Methods employed for the removal of this material from water are coagulation with either sulfuric acid or alum, followed by multiple pond settling. Adjustment of pH is accomplished by addition of either lime or caustic prior to final discharge.

Mine drainage from the single open-pit lode mine is settled prior to discharge. Tailings from the flotation mill in which ore from this mine is processed are collected and settled in an old mining pit. Clarified decant from this pit is recycled to the mill for reuse. Discharge from this pit to a river occurs only seasonally as a result of rainfall and runoff during spring months.

One of the two beach-sand operations located in New Jersey is inactive at present. Recycle of all wastewater for reuse was practiced; consequently, no discharge occurred at this site.

Nickel

A relatively small amount of nickel is mined domestically, all from one mine in Oregon (Mine 6106). This mine is open-pit, and there is a mill at the site, but it only employs physical processing methods. The ore is washed and transmitted to an on-site smelter. Mine and Mill 6106 is profiled in Table III-30. As shown below, production has decreased slightly from 1969 to 1980 (References 5, 18, 20, and 27):

Year	Production Metric Tons	Short Tons
1969	15,483	17,056
1970	14,464	15,933
1971	15,465	17,036
1972	15,309	16,864
1973	16,587	18,272
1974	15,086	16,618
1975	15,421	16,987
1976	14,951	16,469
1977	13,024	14,347
1978	12,263	13,509
1979	13,676	15,065
1980	13,302	14,653

Depending on the outcome of on-going exploration, nickel production may increase in the next 5 to 10 years, and the Bureau of Mines predicts a significant increase in production by 1985. Nickel production is possible both from the Minnesota sulfide ores and from West Coast laterite deposits similar to (but lower in grade than) the deposit presently worked at Riddle, Oregon. Both cobalt production and nickel recovery from laterite ores may involve an increase in the use of leaching techniques.

Water used in beneficiation and smelting of nickel ore is extensively recycled, both within the mill and from external wastewater treatment processes. Most of the plant water is used in the smelting operation since wet-beneficiation processes are not practiced. Water is used for ore belt washing, for cooling, and for slag granulation in scrubbers or ore driers. Water recycled within the process is treated in two settling ponds which are arranged in series. The first of these, 4.5 hectares (11 acres) in area, receives a process water influx of 12.3 cubic meters (3,256 gallons) per minute, of which 9.9 cubic meters (2,624 gallons) per minute are returned to the process. Overflow to the 5.7 hectare (14 acre) second pond amounts to 1.2 cubic meters (320 gallons) per minute. This second pond also receives runoff water from the open-pit mine site which is highly seasonal, amounting to zero for approximately 3 months, but reaching as high as 67,700 cubic meters (17.9 million gallons) per day during the (winter) rainy season. The lower pond has no surface discharge during the dry season. The inputs are balanced by evaporation and subsurface flow to a nearby creek. A sizeable discharge results from runoff inputs during wet weather. Average discharge volume over the year amounts to 3,520 cubic meters (930,000 gallons) per day.

Vanadium

This subcategory includes facilities which are engaged in the primary recovery of vanadium from non-radioactive ore; however, there is only one active facility in this subcategory, Mine/Mill 6107. The vanadium subcategory is profiled in Table VIII-31.

At vanadium Mine/Mill 6107, vanadium pentoxide, V_2O_5, is obtained from an open-pit mine by a complex hydrometallurgical process involving roasting, leaching, solvent extraction, and precipitation. The process is illustrated in Appendix A. In the mill, a total of 6,200 cubic meters (1.6 million gallons) of water are used in processing 1,270 metric tons (1,400 short tons) of ore. This includes scrubber and cooling wastes and domestic use.

Ore from the mine is ground, mixed with salt, and pelletized. After roasting at 850 C (1562 F) to convert the vanadium values to soluble sodium vanadate, the ore is leached and the solutions are acidified to a pH of 2.5 to 3.5. The resulting sodium decavanadate ($Na_6V_{10}O_{28}$) is concentrated by solvent extraction, and ammonia is added to precipitate ammonium vanadate. This is dried and calcined to yield a V_2O_5 product.

The most significant effluent streams are from leaching and solvent extraction, wet scrubbers or roasters, and ore dryers. Together, these sources account for nearly 70 percent of the effluent stream, and essentially all of its pollutant content. Production of vanadium is summarized below (References 5 and 18):

	Production	
Year	**Metric Tons**	**Short Tons**
1973	3,737	4,117
1974	4,756	5,240
1975	4,731	5,213
1976	7,330	8,076
1977	6,866	7,565
1978	4,036	4,446
1979	5,302	5,841

TABLE III-1. PROFILE OF IRON MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
1101	MN	1957	A	Iron Ore	33,000,000	OP	Settling	To surface	80×10^3
1102	MN	1965	A	↓	8,230,000	↓	↓	To surface	2.6×10^3
1103	MN	1948	A	↓	4,000,000	↓	↓	None	0
1104	MN	1953	A	↓	1,640,000	↓	↓	To surface	4.3×10^3
1105	MN	1967	A	↓	8,300,000	↓	↓	To surface	48×10^3
1106	MN	1967	A	↓	40,000,000	↓	↓	None	0
1107	MI	1959	A	↓	5,300,000	↓	↓	To surface	13.4×10^3
1108	MI	1956	A	↓	8,800,000	↓	↓	To surface	15.8×10^3
1109	MI	1964	A	↓	16,400,000	↓	↓	To tailings pond	na
1110	PA	1958	A	↓	2,600,000	UG	↓	To tailings pond or mill circuit	na
1113	MN	1976	A	↓	25,000,000	OP	↓	To tailings pond or mill circuit	0
1112	MN	1967	A	↓	8,700,000	↓	↓	To surface	26.5×10^3
1111	MN	1955	A	↓	31,000,000	↓	↓	To surface	52.9×10^3
1114	MO	1961	A	↓	2,360,000	UG	↓	To surface	5.4×10^3
1115	MO	1963	I	↓	2,200,000	UG	↓	To surface	na
1116	WI	1967	A	↓	2,200,000	OP	↓	Seepage basin	0
1117	UT	1946	A	↓	2,400,000	↓	↓	To surface	na
1118	CA	1948	A	↓	8,190,000	↓	↓	None	0
1119	WY	1962	A	↓	4,400,000	↓	↓	To surface	1.7×10^3
1120	MI	1974	A	↓	4,200,000	↓	↓	To surface	0.7×10^3

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; U.D. – under development; EXP – exploration underway.

TABLE III-1. PROFILE OF IRON MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility).	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1121	MN	1968	A	Iron Ore	1,083,000	OP	Settling	To surface	22.5×10^3
1122	MN	1973	A		7,400,000			In pit settling and pump to surface	67.4×10^3
1123	MN	1917	A		2,300,000			Same as above	11.3×10^3
1124	MN	1933	A		10,800,000			To surface	11.3×10^3
1125	MN	1965	A		1,400,000			unk	na
1126	MN	1965	A		2,200,000			unk	na
1127	UT	1953	A		1,700,000			None	0
1128	NM	1938	A		65,000			None	0
1129	TX	1947	A		2,160,000			To surface	na
1130	NY	1942	A		1,800,000		unk	unk	na
1131	NY	1944	A		3,500,000		Settling	Surface	1.7×10^3
1132	WY	1900	A		1,300,000	UG	unk	unk	na
1133	MN	1974	I		0	OP	na	na	
1134	MN	1974	A		1,300,000		unk	unk	
1135	MN	1960	A		1,100,000		unk	unk	
1136	MI	1940	I		273,000	UG	Neutralization; Settling	To surface	456×10^3
1137	CA	1971	A		450,000	OP	unk	None	0
1138	MN	1976	A		8,830,000	OP	Settling	None	0

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; U.D. – under development; EXP – exploration underway

TABLE III-1. PROFILE OF IRON MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1139	GA	unk	unk	Iron ore	na	unk	unk	unk	na
1140	MN	unk	↓	↓	↓	↓	↓	↓	↓
1141	MN	unk	↓	↓	↓	↓	↓	↓	↓
1142	MN	1943	A	↓	↓	↓	↓	↓	↓
1143	MI	1957	A	↓	2,400,000	↓	↓	↓	0.5×10^3
1144	MI	1943	A	↓	1,700,000	UG	Settling	To surface	12.3×10^3
1145	NV	1960	A	↓	104,000	OP	↓	↓	2.4×10^3
1146	MN	unk	A	↓	600,000	OP	↓	↓	37.9×10^3
1147	MN	unk	A	↓	375,000	OP	↓	↓	8.3×10^3

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†To convert to daily gallons, multiply values shown by 264.173
Status Code: A – active; I – inactive; S – seasonal; U.D. – under development; EXP – exploration underway

TABLE III-2. PROFILE OF IRON MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1101	MN	1957	A	Iron ore pellets	10,600,000	Magnetic sep.	Settling	None	0
1102	MN	1965	A	Iron ore pellets	2,840,000	Magnetic sep.			0
1103	MN	1948	A	Iron ore	2,200,000	Jig; wash; heavy media		↓	0
1104	MN	1953	A	Iron ore	434,000	Jig; wash; heavy media		To surface	22.5 x 10^3
1105	MN	1967	A	Iron ore pellets	2,600,000	Magnetic sep.		None	0
1106	MN	1967	A		12,250,000	Magnetic sep.		None	0
1107	MI	1959	A		2,045,000	Magnetic sep.; flotation		To surface	10.2 x 10^3
1108	MI	1956	A		3,600,000	Flotation			32.7 x 10^3
1109	MI	1964	A		5,500,000	Magnetic sep.; flotation			23 x 10^3
1110	PA	1958	A		1,160,000	Magnetic sep.; flotation		↓	6.54 x 10^3
1113	MN	1976	A		5,400,000	Magnetic sep.		None	0
1112	MN	1967	A		2,600,000	Magnetic sep.		None	0
1111	MN	1955	A		10,500,000	Magnetic sep.		None	0
1114	MO	1961	A		1,400,000	Magnetic sep.; flotation		To surface	6.5 x 10^3
1115	MO	1968	I	↓	950,000	Magnetic sep.; flotation	↓	↓	16.1 x 10^3

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Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-2. PROFILE OF IRON MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1116	WI	1967	A	Iron ore pellets	685,000	Magnetic sep.	Settling	None	0
1118	CA	1948	A	Iron ore Iron ore pellets	3,050,000	Wash; jig; heavy media; magnetic sep.	↓	None	0
1119	WY	1962	A	Iron ore pellets	1,630,000	Magnetic sep.	↓	To surface	Minimal
1120	MI	1974	A	Iron ore pellets	4,200,000	Selective flocculation	↓	To surface	unk
1117	UT	1946	A	Iron ore	1,200,000	Heavy media; magnetic sep.	↓	To surface	na
1121	MN	1968	A	↓	480,000	Wash	↓	To surface	5.7 x 10^3
1122	MN	1973	A	↓	700,000	Wash; heavy media	↓	None	0
1123	MN	1917	A	↓	602,000	Wash; heavy media; jig	↓	None	0
1124	MN	1933	A	↓	1,080,000	Wash; heavy media	↓	None	0
1125	MN	1965	A	↓	150,000	Wash	↓	To surface	na
1126	MN	1965	A	↓	330,000	Wash; jig; heavy media	↓	↓	↓
1127	UT	1953	A	↓	950,000	Wash	↓	None	0
1128	NM	1938	A	↓	40,000	None	None	None	0
1129	TX	1947	A	Sinter–Iron ore	725,000	Wash	Settling	None	0
1130	NY	1942	A	Iron ore	261,000	Magnetic sep.	unk	unk	na

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TABLE III-2. PROFILE OF IRON MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1131	NY	1944	A	Sinter -Iron ore	943,000	Magnetic sep.	Settling and Filtration	Filtrate to surface	61.8×10^3
1132	WY	1900	A	Iron ore	500,000	Jig; heavy media	unk	unk	na
1133	MN	1974	I	Iron ore	0	–	na	na	closed
1134	MN	1974	A	Iron ore	100,000	Wash	unk	unk	na
1135	MN	1960	A	Iron ore	195,000	Heavy media	unk	unk	na
1138	MN	1976	A	Iron ore pellets	2,640,000	Magnetic sep.	Settling	None	0
1137	CA	1971	A	↓	na	unk		None	0
1139	GA	unk	unk	↓	↓	↓	unk	None	0
1140	MN	unk	unk	↓	↓	↓	↓	unk	na
1141	MN	unk	unk	↓	↓	↓	↓	↓	↓
1142	MN	1943	A	↓	673,000	Gravity	Settling	None	0
1143	MN	1957	I	↓	0	–	unk	To surface	3.5×10^3
1145	NV	1960	A	↓	104,000	None	None	unk	unk
1146	MN	unk	A	↓	337,000	Screen; gravity	↓	None	0
1147	MN	↓	A	↓	182,000	Gravity	↓	↓	0
1148	MN	↓	A	↓	1,430,000	Gravity	↓	↓	0
1149	MN	↓	I	↓	0	Gravity	Settling	To surface	na

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†To convert to daily gallons, multiply values shown by 264.173
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-3. PROFILE OF COPPER MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION [1] (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2101	NV	1910	A	Cu ore Cement Cu	7,196,000 (1973)	OP; Dump leaching	Mine water to leach circuit	None	0
2102	AZ	1961 1973	A	Cu ore Cement Cu	5,457,000 8,200	OP; Vat leaching	Recycle to mill	None	0
2103	NM	1969	A	Cu ore Cement Cu	13,974,000 (1973)	OP; Dump leaching	None	None	0
2104	NM	1913	A	Cu ore Cement Cu	7,349,000 (1973)	OP; Dump leaching	Settling	To surface (intermittent)	680 (average)
2107	AZ	1885	I (mines) A (leach)	Cement Cu	3,994,000 (1972)	OP; UG; Dump leaching	unk	None	0
2108	AZ	1951	A	Cu ore Cement Cu	2,782,000 5,000	OP; Dump leaching	Total recycle	None	0
2109	AZ	1972	A (OP) UD (UG)	Cu ore	3,383,000	OP; UG	Recycle to mill	None	0
2110	AZ	1968	I (OP) A (leach)	Cu ore Cement Cu	3,711,000 (1973)	OP; Dump leaching	Total recycle	None	0
2111	AZ	1962	I	Cu ore	1,480,000 (1973)	OP	Mine water normally used in mill circuit, discharged at present due to inactive status of mine	To surface	NA

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Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-3. PROFILE OF COPPER MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION [1] (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2112	AZ	1974	UD	Cu ore	635,000 (projected)	UG	To be used in mill	None	0
2113	AZ	1917	A	Cu ore	9,381,000 (1973)	OP	Used in mill	None	0
2115	AZ	1910	A	Cu ore	1,411,000	UG	Used in mill	None	0
2116	AZ	1955	A	Cu ore Cement Cu	8,894,000 31,000	OP; Dump, vat leaching	Total recycle of leach circuit. Mine water used as potable water.	None	0
2117	TN	1899	A (UG) UD (OP)	Cu ore	1,836,000	UG; OP	Lime pptn; aeration; settle. Water from inactive mine to tailing pond.	To surface	unk
2118	AZ	1942	A	Cu ore Cement Cu	16,653,000 (1973) na	OP; Dump leaching	Mine water to leach circuit	None	0
2119	AZ	1956	A	Cu ore	13,620,000	UG	Mine water to mill circuit	None	0
2120	MT	1955	A (OP) (leach) I (UG)	Cu ore Cement Cu	15,419,000 16,300 (1973)	OP; UG; Dump leaching	Lime pptn; settle; pH adjustment; partial recycle to mill	To surface	35,960 (Combined d/c) 190 (sep mine water)
2121	MI	1953	A	Cu ore	3,281,000	UG	Settle; alkaline sedimentation; secondary settling	To surface (seasonal)	121,120 (Combined d/c)

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[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-3. PROFILE OF COPPER MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2122	UT	1906	A (OP) (leach)	Cu ore	32,208,000	OP; Dump leaching	Mine water to leach circuit with total recycle	None	0
			UD (UG)	Cement Cu	36,300 (1973)	UG			
2123	AZ	1940	A	Cu ore Cathode Cu	1,854,000 6,620	OP; Dump leach	Mine water to leach circuit - with total recycle	None	0
2124	AZ	1915	A	Cu ore Cathode Cu Cement Cu	6,086,000 (1973) 12,600 (1972) 7,400 (1973)	OP; Dump; heap; vat and in-situ leaching	unk	None	0
2125	AZ	1971	I	Cement Cu	no production in 1976	In-situ leaching	Total recycle	None	0
2126	NV	1953	A (due to close) (in 1978)	Cu ore Cement Cu	7,256,000 na	OP; Vat and dump leaching	Used in mill	None	0
2130	NM	1967	A	Cu ore	Confidential	OP; UG	Used in mill circuit	To surface	combined d/c
2131	NV	1969	UD	Cu ore Cement Cu	na	OP; Heap leaching	unk	None	0
2132	NV	1967	I	Cu ore Cement Cu	na	OP; Dump leaching	unk	None	0
2133	NV	1974	I	Cu ore	0	UG; OP	To tailing pond	None	0

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[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-3. PROFILE OF COPPER MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2134	ID	1972	A	Cu ore	na	OP	No minewater	None	0
2135	AZ	1964	A (OP; heap leach) UD (vat leach)	Cathode Cu	7,500	OP; Heap leaching; Vat leaching	Total recycle	None	0
2136	AZ	unk	I	Cement Cu	na	OP; Dump leaching	unk	None	0
2137	AZ	1974	I	Cu ore Cement Cu	na	OP; Tailings leach	No minewater	None	0
2138	AZ	1976	A	Cu ore Cement Cu Cathode Cu	4,535,000 (design)	UG; Vat leaching	Minewater to mill; leach water evap.	None	0
2139	AZ	1971	A	Cu ore	29,478,000	OP	unk	None	0
2140	AZ	1964	A	Cu ore Cement Cu	5,261,000 2,900	OP; Dump leaching	Used in mill	None	0
2141	AZ	1959	I	Cu ore Cement Cu	4,807,000 na	OP; Dump leaching	unk	None	0
2142	AZ	1975	A	Cu ore Cathode Cu	1,652,000 4,500 (capacity)	Leaching	Total recycle	None	0
2143	AZ	1977	unk	unk	unk	unk	unk	unk	na

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TABLE III-3. PROFILE OF COPPER MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2144	AZ	1970	UD	Cu ore Cement Cu	na na	UG; In-situ leaching	unk	unk	0
2145	AZ	1965	A	Cu ore Cathode Cu	16,926,000 (1972) na	OP; Vat leaching	Used in mill	None	0
2146	AZ	1974	A	Cu ore	na	OP	Used in mill	None	0
2147	AZ	1957	I	Cu ore	17,777,000	OP	Used in mill	None	0
2148	AZ	1954	A (leach) I (OP)	Cu ore Cement Cu	na 1,500	OP; Dump leaching	Leach circuit is totaly recycled	unk	0
2149	AZ	unk	I	Cu ore Cement Cu	na na	OP; Vat leaching	unk	unk	0
2150	UT	1979	UD	Cu ore	unk	UG	unk	unk	unk
2151	MI	unk	EXP	Cu ore	na	UG	unk	unk	na
2154	AZ	unk	EXP	Cu ore	na	UG	unk	unk	na

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TABLE III-3. PROFILE OF COPPER MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
2155	OR	1892	A	Cu, Au, Ag ores	36,300	UG	None	None	0
2156	ID	unk	I	Cu ores	NA	NA	NA	NA	unk

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†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-4. PROFILE OF COPPER MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2101	NV	1910	A	Cu, Mo Conc.	235,000 (1973)	Flotation	Settle in tailings pond. Decant partially recycled to mill; remainder settled in second pond	Second pond overflow used for agricultural irrigation	0
2102	AZ	1961	A	Cu Conc. Mo Conc.	92,000 250	Flotation	Tailings pond w/ recycle of pond decant to mill	None	0
2103	NM	1969	A	Cu Conc.	421,000 (1973)	Flotation	unk	None	0
2104	NM	1913	A	Cu Conc. Mo Conc.	na na	Flotation	Total recycle	To surface (intermittent)	680
2107	AZ	1885	I	unk	na	Flotation	unk	None	0
2108	AZ	1951	A	Cu Conc. Mo Conc.	59,000 na	Flotation	Total recycle	None	0
2109	AZ	1974	A	Cu Conc.	62,000	Flotation	Total recycle	None	0
2111	AZ	1962	I	Cu Conc.	na	Flotation	Total recycle	unk	na
2112	AZ	1978	UD	Cu Conc.	41,000 (projected)	Flotation	Total recycle	None	0
2113	AZ	1924	A	Cu Conc.	164,000 (1973)	Flotation	Total recycle	None	0
2115	AZ	1913	A	Cu Conc.	1,411,000	Flotation	Total recycle	None	0
2116	AZ	1910	A	Cu Conc. Mo Conc.	219,000 682	Flotation	Total recycle	None	0

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Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-4. PROFILE OF COPPER MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
2117	TN	1899	A	Cu Conc. Fe Conc. Zn Conc.	73,000 771,000 9,000	Flotation	Lime pptn; aeration; settling	To surface	32,700 (combined d/c)
2118	AZ	1942	A	Cu Conc.	440,000 (1973)	Flotation	Total recycle	None	0
2119	AZ	1956	A	Cu Conc. Mo Conc.	381,000 2,100	Flotation	Total recycle	None	0
2120	MT	1955	A	Cu Conc.	327,000	Flotation	See Mine Code 2120	To surface	35,960 (combined d/c)
2121	MI	1954	A	Cu Conc. Ag Conc.	125,000 185	Flotation	Lime; pptn; settle; secondary settling; polyelectrolyte addition	To surface	121,120 (combined d/c)
2122	UT	1917	A	Cu Conc. Mo Conc.	742,000 (1973) 10,700 (1973)	Flotation	$FeCl_3$ addition; oxidation; lime pptn; settling	To surface	32,200 (combined d/c)
2123	AZ	1940	A	Cu Conc. Mo Conc. Ag Conc.	9,500 180 1.9	Flotation	Total recycle	None	0
2124	AZ	unk	A	Cu Conc. Mo Conc.	69,000 (1973) na	Flotation	unk	None	0
2126	NV	1953 (due to close in 1978)	A	Cu Conc.	na	Flotation	Total recycle	None	0
2130	NM	1967	A	Cu Conc.	Confidential	Flotation	Partial recycle	To surface	minimal

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[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-4. PROFILE OF COPPER MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION [1] (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
2132	NV	1967	I	Cu Conc. Au Conc. Ag Conc.	na na na	Flotation	unk	unk	0
2133	NV	1975	I	Cu Conc.	0	Flotation	Total evaporation	None	0
2134	ID	1973	A	Cu Conc. Ag Conc.	na na	Flotation	Recycle	unk	na
2137	AZ	1974	I	Cu Conc.	na	Flotation	Total recycle	None	0
2138	AZ	1976	A	Cu Conc.	131,000 (design)	Flotation	Total recycle	None	0
2139	AZ	1971	A	Cu, Mo, Ag Conc.	369,000	Flotation	Total recycle	None	0
2140	AZ	1964	A	Cu Conc. Mo Conc.	87,000 2700	Flotation	Total recycle	None	0
2141	AZ	1959	I	Cu Conc. Mo Conc.	50,800 na	Flotation	Total recycle	None	0
2145	AZ	1969	A	Cu Conc. Mo Conc.	na na	Flotation	Impoundment; recycle planned	None	0
2146	AZ	1974	A	Cu Conc. Mo Conc.	na na	Flotation	Total recycle	None	0
2147	AZ	1957	I	Cu, Mo, Ag Conc.	na na	Flotation	Total recycle	None	0

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[1] Unless otherwise indicated, production data represent 1976 information.

TABLE III-4. PROFILE OF COPPER MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION[1] (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
2148	AZ	1954	I	Cu Conc.	na	Flotation	unk	unk	na
2150	UT	To open in 1979	UD	Cu Conc.	na	unk	unk	unk	na
2151	MI	unk	I	Cu Conc.	na	Flotation	unk	unk	na

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TABLE III-5. PROFILE OF LEAD/ZINC MINES

MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
3101	ME	1972	I	Zn/Cu ore	190,000	UG	Minewater used as mill feed	None	0
3102	MO	1969	A	Pb/Zn ore	1,482,000	UG	Minewater (67%) used as mill feed; remainder to mill wastewater treatment system	To surface	7,570
3103	MO	1969	A	Pb/Zn/Cu ore	972,300	UG	Minewater (62%) used as mill feed, remainder to mill wastewater treatment system	To surface	3,115
3104	NY	1931	A	Zn/Pb ore	1,009,100	UG	Minewater used as millfeed	None	0
3105	MO	1973	A	Pb/Zn/Cu ore	1,032,000	UG	Minewater (25%) used as mill feed; remainder to settling	To surface	8,300
3106	PA	1955	A	Zn ore	347,700	UG	Minewater (5%) used as mill feed. Minewater (95%) combined w/ tailings pond decant to secondary settling	To surface	107,900
3107	ID	1887	A	Zn/Pb ore	709,000	UG	Minewater combined w/ mill tailings, smelter and refinery wastewater for treatment. Backfill mines stopes w/ sand tails.	To surface	22,500 (combined flow)

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†To convert to daily gallons, multiply values shown by 264.173

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TABLE III-5. PROFILE OF LEAD/ZINC MINES (Continued)

MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
3108	TN	1957	A	Zn ore	354,500	UG	Minewater used as mill feed	None	0
3109	MO	1968	A	Pb/Zn ore	1,013,000	UG	Sanitary minewater and mill water pumped to sewage lagoon. Process minewater pumped to mill tailings pond w/ partial recycle	To surface	3,370
3110	NY	1915	A	Zn ore	93,700	UG	Minewater used as mill feed	None	0
3111	TN	1958	I	Zn ore	90,700	UG	Settle in underground sumps.	To surface	3,600
3112	NM	unk	A	Pb/Zn ore	122,400	UG	None	To surface	2,460
3113	CO	1971	A	Pb/Zn ore	184,500	UG	Minewater (50%) used as mill feed. No treatment of excess. Backfill mine w/ mill sand tails.	To surface	6,400
3114	ID	1939	I	Pb/Ag ore	61,600	UG	Minewater to mill treatment system. Sand tails used as minefill.	To surface	95
3115	WI	1950	I	Pb/Zn ore	334,000	UG	Minewater (62%) used as mill feed. Remainder to settling ponds.	To surface	4,160

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-5. PROFILE OF LEAD/ZINC MINES (Continued)

MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
3116	CO	1930	I	Zn/Pb/Cu/ Ag ore	180,000	UG	Lime ppt.	To surface	3,300
3117	VA	1928	A	Zn ore	541,000 (includes production from mine 3118)	UG	Settle in underground mine sumps	To surface	6,280
3118	VA	1928	A	Zn ore	see above	UG	Settle in underground mine sumps	To surface	39,000
3119	MO	1954	A	Pb/Cu ore	586,500	UG	Mill feed (18%); remainder to multiple pond system	To surface	4,920
3120	ID	1950	A	Pb/Zn ore	157,500	UG	Minewater to mill wastewater treatment system. Backfill mine stopes w/ sand tails and cement.	To surface	2,200
3121	ID	1940	A	Pb/Zn/ Ag ore	256,500	UG	Minewater (23%) to mill wastewater treatment systems. Excess receives no treatment. Backfill mine stopes w/ mill sand tails.	To surface	4,700
3122	MO	1967	A	Pb/Zn/ Cu ore	1,008,000	UG	Minewater (9%) used as mill feed; remainder to multiple pond system.	To surface	26,000

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-5. PROFILE OF LEAD/ZINC MINES (Continued)

MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m3 †)
3123	MO	1960	A	Pb/Cu/ Zn ore	1,609,000	UG	Part of minewater settled in multiple pond systems. Part used as mill feed with remainder to to settling pond.	To surface	36,700
3124	NJ	1850	A	Zn ore	186,000	UG	None	To surface	950
3125	NY	1940	A	Zn ore	21,400	UG	None	To surface	1,360
3126	TN	1955	A	Zn ore	682,000	UG	Minewater used in mill circuits as required	To surface	0 - 3,340
3127	TN	1965	A	Zn ore	654,500	UG	Minewater (60%) used as mill feed; remainder receives no treatment.	To surface	5,500
3128	TN	1960	A	Zr ore	477,000	UG	Multiple pond system	To surface	5,450
3129	UT	1890	A (OP) I (UG)	Pb/Zn/Cu/ Ag/Au ore	na	OP; UG	Open pit minewater to adjacent mine (#2122). Underground mine drainage sold for irrigation.	None	0
3130	UT	1975	I	Pb/Zn/Ag ore	na	UG	Lime pptn; floc addition, multiple pond. Mill sand tails for backfill.	To surface	32,700
3131	WI	unk	A	Pb/Zn ore	na	UG	None	To surface	7,600
3132	WI	unk	A	Pb/Zn ore	na	UG	None	To surface	4,400

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-5. PROFILE OF LEAD/ZINC MINES (Continued)

MINES	LOCATION (state)	YEAR OPENED. (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
3133	WI	unk	I	Pb/Zn ore	0	UG	None	To surface	na
3134	WA	unk	I	Pb/Zn ore	273,000 (includes #3135 prod)	UG	Settle in underground sumps	To surface	2,725
3135	WA	unk	I	Pb/Zn ore	(see above)	UG	No mine drainage	None	0
3136	NV	1977	UD	Pb/Zn ore	~ 114,000 (pilot scale)	UG	No mine drainage	None	0
3137	AZ	1968	I	Zn/Cu ore	84,500 (1973)	UG	Minewater used in mill circuit.	None	0
3138	CO	1880	I	Pb/Zn/Cu/ Ag ore	89,100	UG	Portion of minewater to mill. Excess to mill tailings pond w/ evap. and seepage.	To surface (twice/yr)	na
3142	UT	1966	A	Pb/Zn/Ag/ Cd/Au ore	196,000	UG	Portion of minewater to mill. Excess combined w/ tailings pond decant for impoundment w/ evap. and seepage.	None	0
4404	CO	1921	A	Pb/Zn/Cu/ Au/Ag ore	369,100	UG	Portion minewater to mill; excess to impoundment. Other mine portals receive no treatment.	To surface	3,800
3143	CO	1966	A	Pb/Ag ore	~ 54,500	UG	None	To surface	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-6. PROFILE OF LEAD/ZINC MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m3 †)
3101	ME	1972	I	Zn, Cu Conc.	25 600 (1973)	Flotation	Lime pptn; multiple pond, partial recycle	To surface	1,430
3102	MO	1969	A	Pb Conc. Zn Conc.	228,600 41,600	Flotation	Alk. sed.; multiple pond, biological meander	To surface	15,140 (mill) 22,700 (total)
3103	MO	1969	A	Pb Conc. Zn Conc. Cu Conc.	92,400 16,000 9,800	Flotation	Alk. sed.; multiple pond; partial recycle	To surface	9,460 (mill-water to pond) 9,840 (total)
3104	NY	1972	A	Pb, Zn Conc.	113,100	Flotation	Alk. sed.	To surface	6,800
3105	MO	1973	A	Pb Conc. Zn Conc. Cu Conc.	60,800 12,200 45	Flotation	Alk. sed.; total recycle	None	0
3106	PA	1955	A	Zn Conc.	54,500	Flotation	Alk. sed.; multiple pond	To surface	5,680 (mill) 107,900 (total)
3107	ID	1946	A	Pb Conc. Zn Conc. Ag Conc.	24,000 41,500 345	Flotation	Sed.; aeration; flocculation, lime pptn.; clarification; high-density sludge process	To surface	4,353 (mill) 23,650 (total)
3108	TN	1957	A	Zn Conc.	16,100	Flotation	Alk. sed.	To surface	216
3109	MO	1968	A	Pb Conc. Zn Conc.	75,000 11,100	Flotation	Alk. sed.; partial recycle	To surface	3,760 (mill) 28,400 (total)
3110	NY	1932	A	Zn Conc.	11,800	Flotation	Alk. sed.; multiple pond	To surface	990 (mill) 2,650 (total)

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-6. PROFILE OF LEAD/ZINC MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
3113	CO	1971	A	Zn Conc. Pb Conc.	28,200 9,100	Flotation	Alk. sed.; multiple pond; partial recycle	To surface	5,300
3114	ID	1939	I	Pb/Ag Conc.	10,550	Flotation	Alk. sed. w/ flocculant addition	To surface	1,400
3115	WI	1950	I	Pb, Zn Conc.	16,250	Gravity (jigging) Flotation	Alk. sed.; multiple pond	To surface	6,800 (mill) 18,000 (total)
3116	CO	1930	I	Zn Conc. Pb Conc.	na	Flotation	Alk. sed.; solar evap.	To surface	3,480 (total)
3118	VA	1928	A	Zn Conc. Pb Conc.	27,700 3,300	Flotation	Alk. sed.	To surface	2,500 (3 mills-combined flow)
3119	MO	1954	A	Pb Conc. Cu Conc.	25,700 2,400	Flotation	Alk. sed.	To surface	6,750
3120	ID	1950	A	Pb/Ag Conc. Zn Conc.	24,000 940	Flotation	Alk. sed. w/ flocculant addition	To surface	2,040 (mill) 4,730 (total)
3121	ID	1940	A	Pb/Ag Conc. Zn Conc.	16,700 25,450	Flotation	Alk. sed. w/ flocculant addition	To surface	3,330 (mill) 5,980 (total)
3122	MO	1967	A	Pb Conc. Zn Conc.	104,000 6,800	Flotation	Alk. sed. w/ partial recycle	None	0 (mill) 2,600 (total)
3123	MO	1960	A	Pb Conc. Zn Conc. Cu Conc.	91,450 8,000 8,580	Flotation	Alk. sed. w/ total recycle	None	0
3126	TN	1975	A	Zn Conc.	45,800	Heavy media; Flotation	Alk. sed. w/ total recycle	None	0
3127	TN	1965	A	Zn Conc.	27,300	Flotation	Alk. sed.; partial recycle	None	0
3130	UT	1975	A	Pb, Zn, Ag Conc.	Confidential	Flotation	Impoundment solar evap.	None	0
3133	WI	1956	I	Pb, Zn Conc.	na	Flotation	Settling	Surface	2,900

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-6. PROFILE OF LEAD/ZINC MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
3134	WA	1950	I	Pb Conc. Zn/Cd Conc.	na	Flotation	Impoundment; solar evap.	None	0
3136	NV	1977	A	Pb/Ag Conc. Zn Conc.	~118,000 (pilot scale)	Flotation	Impoundment; solar evap. and seepage	None	0
3137	AZ	1968	I	Zn Conc. Cu Conc.	na	Flotation	Impoundment; solar evap. and partial recycle	None	0
3138	CO	1965	I	Pb/Cu/Ag Conc. Zn Conc.	na	Flotation	Impoundment; solar evap.	Intermittent to surface (twice/yr)	na
3139	IL	unk	I	Zn Conc. Pb Conc.	na	Gravity (jig); Flotation	unk	To surface	3,300
3140	NM	1951	A	Pb, Zn Conc.	29,000	Flotation	Impoundment; evap. and seepage	None	0
3141	TN	1913	I	Zn Conc.	47,700	Heavy media; Flotation	Alk. sed.	To abandoned mine	0
3142	UT	1970	A	Pb/Ag/Au Conc. Zn/Cd Conc.	12,000 24,500	Flotation	Impoundment; solar evap. and seepage	None	0
3143	CO	1966	A	Pb Conc. Ag Conc.	na	Flotation	Impoundment; solar evap. and partial recycle; multiple pond	None	0
4404	CO	1921	A	Pb/Ag/Au Conc. Zn Conc. Cu Conc.	9,300 16,400 4,400	Flotation	Impoundment; solar evap. and seepage	To surface	0

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-7. PROFILE OF MISCELLANEOUS LEAD/ZINC MINES

NUMBER OF MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †[1])
2	CO	unk	A	Pb/Zn ore	< 400 (Employ 2 total)	UG	unk	unk	na
3	ID	↓	A	Pb/Zn ore	na (Employ 36 total)	UG	↓	↓	↓
1	OR	↓	A	Pb/Zn ore	na (Employs 7 total)	UG	↓	↓	↓
4	TN	↓	A	Zn ore	1,068,000 (comb. prod.) (Employ 191 total)	UG	↓	↓	↓
1	NM	↓	A	unk	(Employs 4 total)	UG	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Source: MESA files and 1976 EMJ Directory

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-8. PROFILE OF MISCELLANEOUS LEAD/ZINC MILLS

NUMBER OF MILLS	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
1	ID	unk	A	unk	na (Employs 7 total)	unk	unk	unk	na
1	MT	↓	A	↓	na (Employs 3 total)	↓	↓	↓	↓
1	OR	↓	A	↓	na (Employs 7 total)	↓	↓	↓	↓
2	TN	1927 1975	A A	Zn Conc. Zn Conc.	28,200 ~ 614,000	Flotation Heavy media; Flotation	↓	↓	↓
1	WI	unk	I	Pb Conc. Zn Conc.	na	Gravity (jigging); Flotation	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-9. PROFILE OF GOLD MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
4101	NV	1965	A	Au ore	744,300 (1973)	OP	None	None	0
4102	CO	unk	A	Pb, Zn, Au ore	162,000 (1973)	UG	None	To surface	3,788
4104	WA	1937	A	Au ore	49,610 (1974)	UG	Impoundment	Indirect – seepage and mill makeup	144
4105	SD	1877	A	Au ore	1,416,387 (1973)	UG	None	Mill makeup	11,500
4115	UT	1965	A	Au/Ag ore; some copper	131,000	UG	None	To surface	na
4123	NV	1973	A	Au ore	na	OP	None	None	0
4124	NM	unk	S	Au ore	na	Placer	Impoundment; recycle	None	0
4125	CA	unk	I	↓	na	UG and Placer	unk	unk	na
4126	AK	1924	S	↓	612,000 m^3** (1975)	Placer; dredging; Au recovered by gravity + amalgamation	Settling and partial recycle	Recycle; To surface	minimal
4127	AK	unk	S	↓	~ Same as above	Same as above	Settling and partial recycle	Recycle; To surface	minimal
4130	NV	1973	I	↓	~ 1530m^3/day** (seasonal)	Placer; mechanical excavation	Impoundment; and solar evap.	None	0
4154	NM	unk	UD	Au ore	680,000	OP	None	To surface	unk

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
**Where placer mining is employed, productions are given in cubic meters of ore mined.
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-10. PROFILE OF GOLD MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
4119	NV	1975	A	Au/Ag	na	Cyanidation; counter current decantation	Recycle of spent leach sol'n in mill; impoundment of tailings	Small volume discharged from tailings pond to desert floor	~ 10% of total
4120	NV	1977	A	Au/Ag	na	Flotation; cyanidation of flotation concentrate	Impoundment; solar evap and recycle	None	0
4121	NV	1969	I	Au	81,200 troy oz [1] (1974)	Cyanidation; agitation leach; heap leach and Zn pptn	Impoundment and recycle	None	0
4122	NV	1973	I	Au	23,100 troy oz (1974)	Cyanidation; heap leach; carbon adsorption and electrowinning	Recycle	None	0
4128	NV	1975	A	Au/Ag	na	Cyanidation; heap leach; carbon adsorption; electrowinning	Impoundment and recycle	None	0
4129	CO	1964	UD	Au/Ag	Expect ~ 300,000	Flotation	Settling; partial recycle	To surface	na
4131	NV	1976	A	Au/Ag	Expect $60\text{-}80 \times 10^3$ troy oz (Au) $30\text{-}40 \times 10^3$ troy oz (Ag)	Cyanidation; heap leach; carbon adsorption; electrowinning	Impoundment and recycle	None	0
4154	NM	unk	UD	Au	unk	Crushing, cyanide leach, carbon adsorption, electrowinning	Recycle	None	0 (planned)

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
[1] To convert to grams, multiply value shown by 31.1
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-10. PROFILE OF GOLD MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m³ †)
4119	NV	1975	A	Au/Ag	na	Cyanidation; counter current decantation	Recycle of spent leach sol'n in mill; impoundment of tailings	Small volume discharged from tailings pond to desert floor	~ 10% of total
4120	NV	1977	A	Au/Ag	na	Flotation; cyanidation of flotation concentrate	Impoundment; solar evap and recycle	None	0
4121	NV	1969	I	Au	81,200 troy oz[1] (1974)	Cyanidation; agitation leach; heap leach and Zn pptn	Impoundment and recycle	None	0
4122	NV	1973	I	Au	23,100 troy oz (1974)	Cyanidation; heap leach; carbon adsorption and electrowinning	Recycle	None	0
4128	NV	1975	A	Au/Ag	na	Cyanidation; heap leach; carbon adsorption; electrowinning	Impoundment and recycle	None	0
4129	CO	1964	UD	Au/Ag	Expect ~ 300,000	Flotation	Settling; partial recycle	To surface	na
4131	NV	1976	A	Au/Ag	Expect $60\text{-}80 \times 10^3$ troy oz (Au) $30\text{-}40 \times 10^3$ troy oz (Ag)	Cyanidation; heap leach; carbon adsorption; electrowinning	Impoundment and recycle	None	0
4132	NM	unk	UD	Au	unk	Crushing, cyanide leach, carbon adsorption, electrowinning	Recycle	None	0 (planned)

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] To convert to grams, multiply value shown by 31.1

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-11. PROFILE OF MISCELLANEOUS GOLD AND SILVER MINES

NUMBER OF MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE (If active)	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
1	AL	unk	A: 1	Au	na	OP	unk	unk	na
na	AK	↓	A: na S: na	Placer; Au Placer; Ag	9146 troy oz[1] 351 troy oz	Dredging; Hydraulic and/or mechanical excavation	Settling; recycle None at most; settling at some.	Bleed to surface To surface	~ 5-10% of total volume 19,000-57,000
39	AZ	↓	A: 20 I: 19	Au ore Au/Ag ore Ag ore	15,708 0 15,273	OP: 6 UG: 13	unk	unk	na
47	CA	↓	A: 19 I: 28	Au ore Au/Ag ore Ag ore Placer Au Placer Ag	1,805 2,221 90 2,809 troy oz 272 troy oz	OP: 11 UG: 8	↓	↓	↓
48	CO	↓	A: 16 I: 32	Au ore Au/Ag ore Ag ore Placer Au	na na 110,935 226 troy oz	OP: 4 UG: 12	↓	↓	↓
41	ID	↓	A: 5 I: 36	Au ore Au/Ag ore Ag ore Placer Au	587 1,480 51,072 24 troy oz	OP: 1 UG: 4	↓	↓	↓
31	MT	↓	A: 8 I: 23	Au ore Au/Ag ore Ag ore Placer Au Placer Ag	4,020 19,204 26,511 143 troy oz 21 troy oz	OP: 2 UG: 6	↓	↓	↓
36	NV	↓	A: 14 I: 22	Au ore Au/Ag ore Ag ore	1,803,183 11,434 229	OP: 12 UG: 2	Most are dry mines	None	na

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
[1] To convert to grams, multiply value shown by 31.1
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-11. PROFILE OF MISCELLANEOUS GOLD AND SILVER MINES (Continued)

NUMBER OF MINES	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE (If active)	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
19	NM	unk	A: 19	Au ore Au/Ag ore Ag ore	1,403,722 0 0	OP: 8 UG: 11	unk	unk	na
2	NC	↓	A: 2	Au/Ag ore	na	OP: 0 UG: 2	↓	↓	↓
18	OR	↓	A: 1 I: 17	Au ore Au/Ag ore Ag ore	2,610 2,974 47	OP: 0 UG: 1	↓	↓	↓
2	SD	↓	A: 1 I: 1	Au/Ag ore	na	OP: 0 UG: 1	↓	↓	↓
12	UT	↓	A: 6 I: 6	Au/Ag ore	na	OP: 1 UG: 5	↓	↓	↓
1	VA	↓	A: 1	Au/Ag ore	na	OP: 1	↓	↓	↓
16	WA	↓	A: 1 I: 15	Au/Ag ore	na	OP: 0 UG: 1	↓	↓	↓

*To convert to annual short tons multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-12. PROFILE OF MISCELLANEOUS GOLD AND SILVER MILLS

NUMBER OF MILLS	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (troy oz. of metal)¹	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m³ †)
5	AZ	unk	A: 3 I: 2	Au/Ag	na	unk	unk	unk	na
4	CA	↓	A: 2 I: 2	Au/Ag	↓	↓	↓	↓	↓
8	CO	↓	A: 5 I: 3	Au/Ag	↓	↓	↓	↓	↓
1	ID	↓	I	unk	↓	↓	↓	↓	↓
1	MT	↓	I	unk	↓	↓	↓	↓	↓
8	NV	↓	A: 4 I: 4	Au/Ag	↓	↓	↓	↓	↓
1	NM	↓	A	Au/Ag	↓	↓	↓	↓	↓
1	OR	↓	I	unk	↓	↓	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

¹To convert to grams, multiply value shown by 31.1

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-13. PROFILE OF SILVER MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
4401	ID	1947	A	Tetrahedrite ore	164,045 (1976)	UG	Multiple pond settling	To surface	800
4402	CO	1967	A	Ag ore	74,426 (1973)	↓	Mechanical lime ppt.	To surface	2,936
4403	ID	1921	A	Tetrahedrite ore	180,000 (1973)	↓	Settled in mill tailings pond	To surface	3,133
4406	ID	1975	A	Tetrahedrite ore	~ 97,950	↓	None	Mill makeup	0
4407	ID	1976	A	Ag ore	na	unk	unk	unk	na
4408	MT	1974	A	Ag ore	~ 67,500	UG	None	None	0
4409	ID	1920	UD	Ag ore	na	↓	None	To surface	na
4410	ID	1952	UD	Tetrahedrite ore	~ 76,500 (1972)	↓	Settled in mill 4403 tailings pond	To surface	2,727
4411	TX	unk	UD	Ag ore	unk	UG	Settling (planned)	To arroyo tributory of creek	56.8**

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

**Allowable discharge (NPDES permit)

TABLE III-14. PROFILE OF SILVER MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
4403	ID	1921	A	Tetrahedrite conc.; Ag + Cu in pyrite conc.	8,200 (1971)	Flotation	Alk. sed.	Decant to surface	3,133
4401	ID	1947	A	Cu/Ag conc. (tetrahedrite)	4,522 (1973)	Flotation	Alk. sed. (multiple pond)	Decant to surface; recycle, seepage	na
4402	CO	1967	A	Pb/Ag conc.	7,587 (1972)	Flotation Carbon-in-Pulp	Alk. sed.	Recycle	955
4406	ID	1975	A	Cu/Ag conc.	Expect ~ 2,700	Flotation	Alk. sed. (multiple pond)	Decant to surface seepage	na
4407	ID	1976	A	Ag	na	Cyanidation; vat leach; Zn pptn	Settling and recycle	None	na
4411	TX	unk	UD	Ag	unk	Crushing, grinding, cyanidation	Recycle	None	0 (planned)

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE IIII-15 PROFILE OF MOLYBDENUM MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6101	NM	1922	A	Mo ore	5,700,000	OP	None	None	0
6102	CO	1922	A	Mo,W,Sn ore	14,000,000	UG and OP	To mill treatment system		3.8×10^3
6103	CO	1978	A	Mo ore	2,200,000	UG	Settling; flocculants, aeration	To surface	9.1×10^3
6110	ID	1983 (expected)	UD	Mo ore	6.8×10^6 (capacity)	OP	To mill tailing pond	None	0
6111	AK	unk	EXP	Mo ore	na	unk	unk	unk	na
6115	CO	unk	I (old Pb/Zn) EXP	Mo ore	UG	na	None	To surface	na
6165	NV	1980	UD	Mo ore	OP	unk	None	None	0

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

**These ferroalloy operations are not included in the current subcategorization scheme as shown in Table II-1.

TABLE III-16. PROFILE OF MOLYBDENUM MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6101	NM	1922	A	MoS_2 conc.	10,831	Flotation	Tailings pond, flocculation; settling H_2O_2	To surface	11,000
6102	CO	1922	A	MoS_2 conc. W conc. Sn conc.	27,000	Flotation; mag. and grav.	Tailings pond; recycle; I X; chlorin; electro-coagulation; flotation	To surface	11,000
6103	CO	1976	A	MoS_2 conc.	na	Flotation	Tailings pond; recycle	None	0
6110	ID	1983 (expected)	UD	MoS_2 conc.	8170 (capacity)	Crushing, concentration	Tailing pond, recycle	None	0
6165	NV	1980	UD	MoS_2, MoO_3 conc. Cu conc.	5447 908	Flotation	Evaporation, and recycle	None	0

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-17. PROFILE OF ALUMINUM ORE MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
5101	AR	1942	A	Bauxite ore	900,000	OP UG (inactive)	Lime neut; settling	To surface	7,970
5102	AR	1899	A	Bauxite ore	~ 900,000	OP	Lime neut.; settling	To surface	32,000

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-18. PROFILE OF TUNGSTEN MINES (PRODUCTION GREATER THAN 5000 MT ORE/YEAR)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6104	CA	1941	A	W,Mo,Cu ore	6.4×10^6	UG	Flocculants; clarification	To mill process and to surface	3.3×10^4
6105	NV	1947	A	W ore	1×10^4	UG	None	To dry wash	<4
6108	NV	1977	A	W ore	na	UG	None	None	0
6109	CA	unk	A	W ore	1.5×10^4	UG	None	On land	<6
6112	NC	unk	I	W ore	3×10^5 (capacity)	UG	None	To surface	na
6117	NV	unk	A	W ore	na	UG and OP	unk	unk	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-19. PROFILE OF TUNGSTEN MINES (PRODUCTION LESS THAN 5000 MT ORE/YEAR)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6119	CO	unk	EXP	W ore	(Expect 10-15)	UG	None	None	0
6120	UT		I	W ore	na	OP and UG	None	None	0
6121	NV		UD	W ore	na	UG	None	None	0
6122	CA		A	W ore	~ 1000	UG	None	None	0
6123	ID		I	W ore	na	UG	unk	unk	na
6126	ID		I[1]	W ore		UG			
6127	ID		I, EXP	W ore		unk			
6128	CA		I	W ore		unk			
6129	NV		A[1]	W ore		UG			
6130	NV	↓	I[1]	W ore	↓	UG	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Best available information; not contacted since July 1977

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-19. PROFILE OF TUNGSTEN MINES (PRODUCTION LESS THAN 5000 MT ORE/YEAR) (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m³ †)
6131	CA	unk	I[1]	W ore	na	UG	unk	unk	na
6132	UT		I[1]			UG			
6133	MT		A[1]			OP	↓	↓	↓
6134	ID		I[1]			UG	None	None	0
6135	CA		A[1]			OP	unk	unk	na
6136	UT		A[1]			UG			
6137	UT		I			UG			
6138	CA		unk			unk			
6139	CA								
6140	CA								
6141	CA								
6142	CA								
6143	CA								
6144	CA		↓		↓		↓	↓	↓
6145	CA		A[1]		≤ 100		None	None	0
6146	CA		unk		~ 500		unk	unk	na
6147	CA		I		< 100		None	To surface	na
6148	NV		I[1]		na	↓	unk	To surface	~ 8,000
6149	ID		I[1]			UG	None	None	
6150	ID		I		↓	unk	unk	unk	na
6151	MT		A[1]		~ 200	unk	None	None	0
6152	OR	↓	A[1]	↓	~ 1000	UG	None	To surface	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Best available information; not contacted since July 1977.

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-20. PROFILE OF TUNGSTEN MILLS (PRODUCTION LESS THAN 5000 MT/YEAR)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6119	CO	unk	UD	W conc	~ 40	Flotation	Recycle planned	Backfill old mine workings	na
6130	NV	↓	I[1]	↓	na	unk	unk	unk	↓
6131	CA	↓	I[1]	↓	na	unk	↓	↓	↓
6132	UT	↓	I[1]	↓	na	unk	↓	↓	↓
6134	ID	↓	I[1]	↓	Very small	Gravity	↓	↓	0
6135	CA	↓	A[1]	↓	na	unk	↓	↓	na
6145	CA	↓	A[1]	↓	≤ 2	Gravity	Recycle	↓	0
6146	CA	↓		↓	~ 6	unk	unk	↓	na
6147	CA	↓	I UD	↓	na	↓	↓	↓	↓
6148	NV	↓	I[1]	↓	na	↓	↓	↓	↓
6149	ID	↓	I[1]	↓	na	Gravity	Settling pond	To surface	↓
6151	MT	↓	A[1]	↓	~ 2	Gravity	unk	To sink hole	~ 5 (est)
6152	OR	↓	A[1]	↓	~ 10	Gravity	Evap. and settling pond	None	0
6153	NV	↓	I[1]	↓	na	unk	unk	unk	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Best available information; not contacted since July 1977.

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-20. PROFILE OF TUNGSTEN MILLS (PRODUCTION LESS THAN 5000 MT/YEAR) (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6109	CA	unk		W conc	~500 (est)	Gravity	Tailings Pond; recycle	None	0
6124	CA	1978 (planned)	UD	W conc	~500 (est)	Gravity (centrifuge)	Tailings pond; recycle	None	0
6159	NV	1974	I (restart planned 1978)	W conc	~100 (est)	Gravity (tables)	Tailings pond; evap.	None	0
6163	CA	unk	I (last report Pb, Ag, Zn prod.)	W conc (from tails)	~50 (est)	Gravity (tables)	Tailings pond; recycle	None	0
6154	MT	unk	A^1	W conc	na	unk	unk	unk	na
6155	NV	unk	A^1	W conc	~25	Gravity	Settling pond; recycle	None	0
6156	UT	unk	A	W conc	na	unk	unk	unk	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-21. PROFILE OF TUNGSTEN MILLS (PRODUCTION GREATER THAN 5000 MT/YEAR)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
6104	CA	1941	A	W conc MoS_2 conc. Cu conc.	39,700 118 120	Flotation	Lime; tailings pond	None	0
6105	NV	1947	A.	W conc.	110	Flotation; grav.	None	To dry wash	~ 200
6108	NV	1977	A	W conc.	1,500 (est)	Flotation	Impoundment	None	0
6112	NC	unk	I	W conc.	na	Flotation	Tailings ponds	unk	unk
6117	NV	↓	A	W conc.	na	Flotation	Tailings pond; recycle	None	0
6157	NV	↓	A	W conc.	~ 50	Flotation	Tailings pond; recycle	None	0
6158	CA	↓	I[1]	W conc.	450 (capacity)	Flotation	Tailings pond; evap.	None	0

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Best available information; not contacted since July 1977.

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-22. PROFILE OF MERCURY MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME $(m^3\dagger)$
9201	CA	1970	I	Hg ore	27,000	OP	None	None	0
9202	NV	1975	A	Hg ore	222,000	OP	None	None	0
(1 mine)	NV	unk	A	Hg ore	na	unk	unk	unk	na
(2 mines)	CA	unk	S	Hg ore	na	unk	unk	unk	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

unk = unknown

OP = Open-Pit

TABLE III-23. PROFILE OF MERCURY MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9201	CA	1970	I	Hg	—	Gravity	Flocculants, settling pond; recycle	Recycle when active; intermittent to surface when inactive	541 Variable (depends on precip)
9202	NV	1974	A	Hg	1,049 (1980)	Flotation	Impoundment; solar evap; partial recycle	None	0

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-24. PROFILE OF URANIUM MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9401	NM	unk	A	Uran. ore + leachate	680,000	UG; in situ (H_2O)	UIX; settling; 90% recycle	To surface	3,200
9402	NM	1970	A	Uran. ore	1,590,000	UG	7 mines: U238 IX; Impoundment; solar evap.	None	0
							2 mines: flocculation; UIX; $BaCl_2$ Co-pptn; settling	To surface	11,400
9404	NM	unk	A		680,000	OP	Impoundment; solar evap.	None	0
9404	NM	1976	A		215,000	UG	Impoundment; solar evap.	None	0
9411	WY	1963	A		324,000	OP	$BaCl_2$ Co-pptn; settling	To surface	13,600
9412	WY	1957	A		510,000	OP	None	None	0
9419	TX	1973	A		549,000	OP	None	To surface	270
9408	CO	1957	A		45,400	UG	Flocculation; $BaCl_2$ Co-pptn; settling	To surface	2.1
9409	WY	1972	A		454,000	OP	None	To surface	1,890
9410	WY	unk	I		0	OP	None	None	8,700
9437	NM	unk	A		87,100	UG	$BaCl_2$ Co-pptn; settling	To surface	18,900
9445	NM	1976	A		326,000	UG	Settling; recycle to mill	None	0
9447	UT	1972	A		238,000	UG	Alum. pptn; $BaCl_2$ Co-pptn; settling	To surface	1.5

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; U.D. – under development; EXP – exploration underway

TABLE III-24. PROFILE OF URANIUM MINES (Continued)

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9448	NM	1976	A	Uran. Ore	Proprietary	OP and UG	None	None	0
9405	CO	unk	A	↓	< 296,000	UG	None	None	0
9450	WY	unk	A	↓	272,000	OP	None	None	0
9452	NM	1969	A	↓	454,000	UG	Flocculation; settling; $BaCl_2$ Co-pptn; U IX; Ra226 IX	To surface	6,540
9413	WY	1957	A	↓	540,000	UG	$BaCl_2$ Co-pptn; settling	To surface	1,360
9455	WA	unk	A	↓	635,000	OP	Impoundment; solar evap.	None	0
9460	WY	1977	A	↓	272,000	OP	$BaCl_2$ Co-pptn; Settling	To surface	3,300
9451	NM	1969	A	Uran. leachate		In situ (H_2O)	Settling; UIX; recycle	None	0
9402	NM	unk	UD	Uran. Ore	na	UG	unk	unk	na
9433	NM	↓	UD	↓	↓	UG	↓	↓	↓
9438	NM	↓	UD	↓	↓	unk	↓	↓	↓
9439	NM	↓	UD	↓	↓	↓	↓	↓	↓
9440	WY	↓	UD	↓	↓	↓	↓	↓	↓
9441	NM	↓	UD	↓	↓	↓	↓	↓	↓
9443	NM	↓	UD	↓	↓	↓	↓	↓	5,500
9444	UT	↓	UD	↓	↓	↓	↓	↓	na
9446	NM	↓	UD	↓	↓	↓	↓	↓	na
9449	WY	↓	UD	↓	450,000	OP	Flocculation; UIX; $BaCl_2$ Co-pptn; settling	To surface	37,300

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-25. PROFILE OF URANIUM MILLS

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9401	NM	unk	A	U_3O_8 Vanad. conc.	2,000[1] na	Alk. leach	Impoundment; solar evap.	None	0
9402	NM	1958	A	U_3O_8 Mo conc.	10,600 40 (1973)	Acid leach	Impoundment; solar evap.	↓	↓
9403	UT	1956	A	U_3O_8 Copper conc. Vanad. conc.	470 159 (1973) na	Acid & Alk. leach	Impoundment; $BaCl_2$ co-pptn; solar evap.; recycle	↓	↓
9404	NM	1955	A	U_3O_8	2,350	Acid leach	Impoundment; solar evap; deep well injection	↓	↓
9411	WY	1971	A	↓	844	Acid leach	Impoundment; solar evap.	↓	↓
9407	WY	1957	A	↓	952	Acid leach	Impoundment; solar evap.	↓	↓
9419	TX	1973	A	↓	716	Acid leach	Settling; recycle	↓	↓
9422	CO	unk	A	↓	857	Alk. leach	Settling; recycle	↓	↓
9423	WA	1978	A	↓	286[1]	Acid leach	Impoundment, evap., lime ppt, recycle	↓	↓
9425	WY	1972	A	↓	1180	Acid leach	Impoundment; solar evap.	↓	↓
9427	WY	unk	A	↓	543[1]	Acid leach	Impoundment; solar evap.	↓	↓
9430	UT	unk	UD	↓	na	unk	unk	unk	na

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] 0.18% U_3O_8 in ore assumed

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-25. PROFILE OF URANIUM MILLS (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9442	WY	1961	I	U_3O_8	0	Acid leach	Impoundment; solar evap.	None	0
9445	NM	1976	A	U_3O_8	882[1]	Acid leach	Recycle; impoundment; solar evap.	None	0
9447	UT	1972	A	U_3O_8	454	Alk. leach	Settling; recycle	None	0
9405	CO	1952	A	U_3O_8 Vanad. conc.	622 3,630	Acid leach	Settling; recycle; solar evap.; flocculation; $BaCl_2$ Co-pptn	To surface	2,190
9450	WY	unk	A	U_3O_8	686[1]	Acid leach	Impoundment; solar evap.	None	0
9452	NM	1977	A	U_3O_8	1,710[1]	Acid leach	Impoundment; solar evap.	None	0
9413	WY	1957	A	U_3O_8	971[1]	Acid leach	Impoundment; solar evap.	None	0
9456	WA	1978	A	U_3O_8	1,140[1]	Acid; SX	Lime ppt, $BaCl_2$, recycle	None	0
9460	WY	1977	A	U_3O_8	444	Acid leach	Recycle; Impoundment; solar evap.	None	0
9449	WY	unk	UD	U_3O_8	410	Acid & heap leach	Impoundment; recycle; solar evaporation	None	0
9472	UT	1980	A	U_3O_8 Vanad. conc.	~ 770 ~ 2,700	Acid leach; SX	Settling; evaporation in lined ponds	None	0

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] 0.18% U_3O_8

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-26. PROFILE OF URANIUM (IN-SITU LEACH) MINES

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate)[1]	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9417	TX	1975	A	Uran. Leachate	113	Ammonia in-situ leach	Recycle + deep well injection	None	0
9424	TX	1976	A	↓	68	Ammonia in-situ leach; U238 IX	↓	↓	↓
9458	TX	1976	A	↓	45	Ammonia in-situ leach	↓	↓	↓
9459	TX	1977	A	↓	227	↓	↓	↓	↓
9461	TX	↓	A	↓	82	↓	↓	↓	↓
9462	TX	↓	A	↓	136	↓	↓	↓	↓
9463	TX	↓	A	↓	na	↓	↓	↓	↓
9464	TX	↓	A	↓	↓	↓	↓	↓	↓
9465	TX	↓	UD	↓	↓	↓	↓	↓	↓
9466	TX	unk	UD	↓	↓	↓	↓	↓	↓
9467	TX	unk	UD	↓	↓	↓	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173
[1] Production given in terms of yellowcake.
Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-26. PROFILE OF URANIUM (IN-SITU LEACH) MINES (Continued)

MILL	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of concentrate) 1	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9468	TX	unk	UD	Uran. leachate	na	Ammonia in-situ leach	Recycle + deep well injection	None	0
9469	TX	↓	↓	↓	↓	↓	↓	↓	↓
9470	TX	↓	↓	↓	↓	↓	↓	↓	↓
9471	TX	↓	↓	↓	↓	Tailings leach	↓	↓	↓

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

[1] Production given in terms of yellowcake

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-27. PROFILE OF ANTIMONY SUBCATEGORY

FACILITY	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons*)	CONCENTRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
Mine 9901	MT	1969	A	Sb ore	18,347 (1979)	UG	None	None	0
Mill 9901	MT	1970	A	$NaSbO_3$	272 as antimony (1979)	Froth flotation; caustic leaching	Impoundment	None	0

*To convert to annual short tons, multiple values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploraton underway

TABLE III-28. PROFILE OF TITANIUM MINES

MINE	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons* of ore mined)	TYPE OF MINE	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
9905	NY	1943	A	Ilmenite ore	900,000	OP	Settling	To surface	2,652
9906	FL	1949	A	Ilmenite (containing beach sands)	7,243,000	Beach placer dredging	See Mill 9906	→	→
9907	FL	1954	A	↓	7,243,000	↓	See Mill 9907	→	→
9908	FL	1971	A	↓	na	↓	See Mill 9908	→	→
9909	FL	1975	I	↓	na	↓	See Mill 9909	→	→
9910	NJ	1973	A	↓	6,585,000	↓	See Mill 9910	→	→
9911	NJ	1962	I	↓	na	↓	See Mill 9911	→	→

*To convert to annual short tons, multiply values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-29. PROFILE OF TITANIUM DREDGE MILLS

MINE	LOCATION (State)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCTS	ANNUAL PRODUCTION* (metric tons of concentrate)†	CONCEN-TRATION PROCESS USED	WASTEWATER TREATMENT TECHNOLOGY	AVERAGE DAILY DISCHARGE VOLUME†† (m^3/day)
9905	NY	1943	A	Ilmenite Magnetite	~200,000 ~130,000	Flotation	Settling; partial recycle	Variable (depends on precip.)
9906	FL	1948	A	Ilmenite Rutile Zircon Staurolite	176,885	Dredging; Wet Gravity Separation; Magnetic and Electrostatic Separation	Acid addition; multiple pond settling; lime addition and secondary settling	25,927
9907	FL	1955	A	↓	126,980	↓	↓	6,170
9908	FL	1971	A	Ilmenite Rutile Zircon Leucoxine Monozite	~45,000 ~23,000 ~23,000	↓	Coagulation with alum or acid; multiple pond settling; neutralized with caustic solution	13,626
9909	FL	1975	I closed October 1979	Ilmenite Zircon Monozite	0	↓	Coagulation with alum; multiple pond settling; neutralized with caustic solution	4,883
9910	NJ	1973	A	Ilmenite	165,000	↓	pH adjustment with alum or lime settling	14,269
9911	NJ	1962	I	Ilmenite Sand and Gravel	0	↓	Settling ponds with recycle of all process water	0

† To convert to annual short tons, multiple values shown by 1.10231

††To convert to daily gallons, multiple values shown by 264.173

Status Code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

*Annual production based on 1978 production

TABLE III-30. PROFILE OF NICKEL SUBCATEGORY

FACILITY	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons*)	PROCESS	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3†)
Mine 6106	OR	1954	A	Ni ore	3.4×10^6 ore	OP	Settling ponds	To mill/smelter treatment system	
Mill 6106	OR	1954	A	Ferronickel	21,050	Screen; crush	Multiple pond settling; partial recycle	To stream	3,500 (seasonal)

*To convert to annual short tons, multiply values shown by 1.10231
†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploration underway

TABLE III-31. PROFILE OF VANADIUM SUBCATEGORY

FACILITY	LOCATION (state)	YEAR OPENED (original facility)	STATUS OF OPERATION	PRODUCT	ANNUAL PRODUCTION (metric tons*)	PROCESS	WASTEWATER TREATMENT TECHNOLOGIES USED	DISCHARGE METHOD	DAILY DISCHARGE VOLUME (m^3 †)
Mine 6107	AR	1966	A	Vanadium ore	363,000	OP	Lime neut. and settling	To surface	6,140
Mill 6107	AR	1967	A	V_2O_5	4,500	Roasting; leaching; solvent extraction; precipitation	Settling and pH adjustment	To surface; multiple discharges	4,460

*To convert to annual short tons, multiple values shown by 1.10231

†To convert to daily gallons, multiply values shown by 264.173

Status code: A – active; I – inactive; S – seasonal; UD – under development; EXP – exploraton underway

SECTION IV

INDUSTRY SUBCATEGORIZATION

During development of effluent limitations and new source standards of performance for the ore mining and dressing category, consideration was given to whether uniform and equitable guidelines could be applied to the industry as a whole, or whether different effluent limitations ought to be established for various subparts of the industry. The ore mining and dressing industry is diverse; it contains nine major SIC codes and ores of 23 separate metals (counting rare earths as a single metal). The wastewaters produced vary in quantity and quality, and treatment technologies affect the economics of each operation differently.

Because this category is complicated, concise descriptions of potential subcategories are necessary to avoid confusion. Therefore, the following definitions are given:

"Mine" is an active mining area, including all land and property placed on, under or above the surface of such land, used in or resulting from the work of extracting metal ore from its natural deposits by any means or method, including secondary recovery of metal ore from refuse or other storage piles derived from the mining, cleaning, or concentration of metal ores.

"Mill" is a preparation facility within which the mineral or metal ore is cleaned, concentrated or otherwise processed prior to shipping to the consumer, refiner, smelter or manufacturer. A mill includes all ancillary operations and structures necessary for the cleaning, concentrating or other processing of the metal ore such as ore and gangue storage areas, and loading facilities.

"Complex" is a facility where wastewater resulting from mine drainage and/or mill processes is combined with wastewater from a smelter and/or refinery operation and treated in a common wastewater treatment system.

FACTORS INFLUENCING SELECTION OF SUBCATEGORIES

The factors that were examined as a possible basis for subcategorization are:

1. Designation as a mine or mill
2. General geologic setting
3. Type of mine (e.g., surface or underground)
4. Ore mineralogy
5. Type of mill process (beneficiation, extraction process)
6. Wastes generated
7. End product
8. Climate, rainfall, and location

9. Reagent use
10. Water use or water balance
11. Treatment technologies
12. Topography
13. Facility age

These factors have been examined to determine if the BPT subcategorization should be retained or if any modification would be appropriate.

Designation as a Mine or Mill

It is often desirable to consider mine water and mill process water separately. Many mining operations do not have an associated mill and deliver ore to a mill located some distance away which other mines also use. In many instances, it is advantageous to separate mine water from mill process wastewater because of differing water quality, flow rate, or treatability. Levels of pollutants in mine waters are often lower or less complex than those in mill process wastewaters. For many mine/mill operations, it is more economical to treat mine water separately from mill water, especially if the mine water requires minimum treatability. Mine water contact with finely divided ores (especially oxidized ores) is minimal and mine water is not exposed to the process reagents often added in milling. Wastewater volume reduction from a mine is seldom a viable option whereas the technology is available to reduce or eliminate discharge from many milling operations. Therefore, development of treatment alternatives and guidelines may be difficult for mines and mills.

Because many operations follow this approach, designation as a mine or mill provides an appropriate basis to classify the industry within subcategories.

That is not to say that mine water and mill water might not be advantageously handled together. In some instances, use of the mine wastewater as mill process water will result in an improved discharge quality because of interactions of the process chemicals and the mine water pollutants.

General Geologic Setting

The general geologic setting (e.g., shape of deposit, proximity to surface) determines the type of mine (i.e., underground, surface or open-pit, placer, etc.). Therefore, geologic setting is not considered a basis for subcategorization.

Type of Mine

The choice of mining method is determined by the general geology; ore grade, size, configuration, and depth; and associated overburden of the ore body. Because no significant differences resulted from application of mine water control and treatment technologies from either surface (open-pit) or underground mines, mine type was not selected as a suitable basis for general subcategorization in the industry.

Ore Mineralogy, Type of Beneficiation Process and Wastes Generated

The mineralogy of the ore often determines the beneficiation process to be used. Both of these factors, in turn, often determine the characteristics of the waste stream, the treatment technologies to be employed, and the effectiveness of a particular treatment method. For these reasons, both ore mineralogy and type of beneficiation processes are important factors bearing on subcategorization. For example, pollutants associated with uranium mining and milling, such as radium 226 and uranium, require treatment technologies not applicable to lead, zinc, and copper facilities. Chemical reagents used in froth flotation processes at lead, zinc, copper and other metals facilities often contain cyanide and other pollutants which are not used in the uranium mills.

On the other hand, many metals are often found in conjunction with one another, and are recovered from the same ore body through similar beneficiation processes. As a consequence, in these instances wastewater treatment technologies and the effectiveness of particular treatment methods will be similar, and, therefore, one subcategory is justified. This is the case, for example, with respect to the copper, lead, zinc, gold, silver, platinum and molybdenum ores subcategory, where several metals often occur in conjunction with each other and are recovered by the froth flotation process. The methods for controlling these metals (and commonly used reagents such as cyanide) in the wastewater discharge is similar throughout the subcategory, and establishing uniform effluent limitations for these facilities is appropriate. In either case, treatment of total suspended solids (i.e., settling) is similar.

Processing (or beneficiation) of ores in the ore mining and dressing industry includes crude hand methods, gravity separation, froth flotation using reagents, chemical extraction, and hydrometallurgy. Physical processing using water, such as gravity separation, discharge the suspended solids generated from washing, dredging, crushing, or grinding. The exposure of finely divided ore and gangue to water also leads to solution of some material. The dissolved and suspended metals content varies with the ore being processed, but wastewater treatments are similar.

Froth flotation methods affect character of mill effluent in several ways. Generally, pH is adjusted to increase flotation efficiency. This and the finer ore grind (generally finer than for physical processing) may have the secondary effect of substantially increasing the solubility of ore components. Reagents used in the flotation processes include major pollutants.

Cyanide and phenol compounds, for example, are used in several flotation processes. Although their usage is usually low, their presence in effluent streams have potentially harmful effects.

Ore leaching operations differ from physical processing and flotation plants. The use of large quantities of reagents such as strong acids and bases and the deliberate solubilization of ore components (resulting in higher percent soluble metals content) characterizes these operations. Therefore, the characteristics of the wastewater quality as well as the treatment and control technologies employed are different than for physical processing and flotation wastewater.

The wastes generated as part of mining and beneficiating metal ores are highly dependent upon mineralogy and processes employed. This was considered in all subcategories.

End Product

The end products are closely allied to the mineralogy of the ores exploited; therefore, mineralogy and processing were found to be more advantageous methods of subcategorization.

Climate, Rainfall, and Location

There is a wide diversity of yearly climatic variations in the United States. Unlike many other industries, mining and associated milling operations cannot choose to locate in areas which have desirable characteristics. Some mills and mines are located in arid regions of the country and can use evaporation to reduce effluent discharge quantity. Other facilities are located in areas of net positive precipitation and high runoff conditions. Treatment of large volumes of water by evaporation in many areas of the U.S. cannot be used where topographic conditions limit space and provide excess surface drainage water. A climate which provides icing conditions on ponds will also make control of excess water more difficult than in a semi-arid area. Climate, rainfall, and location were, therefore, considered in determining whether a particular subcategory can achieve zero discharge.

Reagent Use

Reagent use in many segments of the industry (for example, in the cyanidation process for gold) can potentially affect the quality of wastewater. However, the types and quantities of reagents are a function of the mineralogy of the ore and extraction processes employed. Reagent use, therefore, was included in the consideration of beneficiation processes (e.g., cyanidation for gold).

Water Use and/or Water Balance

Water use or water balance is highly dependent on the choice of process employed or process requirements, routing of mine waters to a mill treatment system or discharge, and potential for use of water for recycle in a process. Beneficiation processes play a determining role in mill water balance and, therefore, water use and/or water balance are considered with beneficiation processes.

Treatment Technologies

Many mining and milling establishments use a single type of effluent treatment method. Treatment procedures vary within the industry, but widespread adoption of differing technologies is not prevalent. Therefore, it was determined that ore mineralogy, mill process and waste characterization provide a more acceptable basis for subcategorization than treatment technology.

Topography

Topographical differences between areas are beyond the control of mine or mill operators, and these place constraints on the treatment technologies employed. One example is tailing pond location. Topographical variations can cause serious problems with respect to rainfall accumulation and runoff from steep slopes. Topography varies widely from one area to another and, therefore, is not a practical basis for subcategorization for national regulations. However, topography is known to influence the treatment and control technologies employed and the water flow within the mine/mill facility. While not used for subcategorization, topography has been considered in the determination of effluent limits for each subcategory.

Facility Age

Many mines and mills have operated for the past 100 years. For mining operations, installation of replacement equipment results in minimal differences in water quality. Many mill processes for concentrating ores in the industry have not changed considerably (e.g., froth flotation, gravity separation, grinding and crushing), but improvements in reagent use, monitoring and control have resulted in improved recovery or the extraction of values from lower grade ores. New and innovative technologies have resulted in changes in the character of the wastes. This is not a function of the age of the facilities, but is a function of extractive metallurgy and process changes. Virtually every facility continuously updates in-plant processing and flow schemes, even though basic processing may remain the same. In addition, most treatment systems employ end-of-pipe techniques which can be installed in either old or new plants. Therefore, age of a facility is not a useful factor for subcategorization in the industry.

SUBCATEGORIZATION

After a review of BPT subcategorization, and after a 36-month data collection effort, it has been determined that the BPT subcategorization (with minor modifications for BAT) adequately represents the inherent differences in the industry. The subcategories are presented in Table IV-1. The proposed subcategorization is based on:

1. Metal being extracted (referred to as Subcategory)
2. Differences between mine and mill wastewater (referred to as Subdivision)
3. Type of mill process (referred to as subpart).

The Settlement Agreement approved by the U. S. District Court for the District of Columbia requires the EPA to establish standards for toxic pollutants and to review the best available technology economically achievable (BAT) for existing sources in the ore mining and dressing industry. The Settlement Agreement, refers to the ore mining and dressing industry as major group 10, as is defined in the Standard Industrial Classification Manual (SIC). Each of the SIC codes in major group 10 were examined in reviewing potential subcategorization using factors required by the Act as contained in this section. The prominent factors identified are: the difference between mine and mills; the ore type, which is generally related to the SIC code; the size of facility (in tonnage); and most importantly, the wastewater characteristics (pollutants found and the treatment employed for removal of the pollutants).

All subcategories are subdivided according to mine drainage and discharge from mills. Subcategories relating to the SIC codes include: iron ore, aluminum ore, uranium ores, mercury ores, titanium and antimony ores (the only representative of metal ores not elsewhere classified SIC 1099). Molybdenum, nickel and tungsten have been separated from Ferroalloys--SIC code 1061. Nickel is a separate subcategory and molybdenum is combined with several other metals.

Because of the similarity of the wastewater discharge from mills and mine drainage, a large subcategory is maintained which is applicable to ores mined or milled for the recovery of copper, lead, zinc, gold, silver and platinum. Molybdenum was also added to this group because of similarity in mill processes. The mine drainage from this subcategory was identified as being of similar pH with relatively high concentrations of heavy metals regardless of the ore mined. The most commonly used mill process in the subcategory is the froth flotation process. In this process, similar reagents are used, and the wastewater from the mills is characterized by high levels of total suspended solids and concentrations of heavy metals. Cyanide is generally used in the mill processes. Many mills in this large subcategory produce two or more metal ore concentrates. Because of the similarity of the wastewater generated, the same wastewater treatment technologies are applicable in this subcategory regardless of the type of ore mined and milled. Because of these factors, the subcategory formerly (in BPT) called Base and Precious Metals is expanded and renamed the Copper, Lead, Zinc, Gold, Silver, Platinum and Molybdenum Ores Subcategory.

COMPLEXES

The subcategorization scheme has subdivisions for mines and mills; complexes are not included. Because of the individuality of complexes, regulation of them has been delegated to the Agency's Regional offices and that practice will continue. As discussed in Section V, Sampling and Analysis Methods, several complexes have been sampled during BAT guideline development and a separate guidance document has been prepared to aid the Regions in preparation of permits commensurate with BAT.

TABLE IV-1. PROPOSED SUBCATEGORIZATION FOR BAT – ORE MINING AND DRESSING

SUBCATEGORY	SUBDIVISION	PROCESS
Iron Ore	Mine Drainage	
	Mills	Physical and/or Chemical Beneficiation
		Physical Beneficiation Only (Mesabi Range)
Copper, Lead, Zinc, Gold, Silver, Platinum, Molybdenum Ores	Mine Drainage	
	Mills or Hydro-metallurgical Beneficiation	Cyanidation or Amalgamation
		Heap, Vat, Dump, In-Situ Leaching (Cu)
		Froth Flotation
		Gravity Separation Methods (incl. Dredge, Placer, or other physical separation methods; Mine Drainage or mines and mills)
Aluminum Ore	Mine Drainage	
Tungsten Ore	Mine Drainage	
	Mills	
Nickel Ore	Mine Drainage	
	Mills	(Physical Processes)
Vanadium Ore*	Mine Drainage	
	Mills	Ore Leaching
Mercury Ore	Mine Drainage	
	Mills	Gravity Separation, Froth Flotation, Other Methods
Uranium Ores	Mine Drainage	
	Mills, Mines and Mills or In-Situ Mines	
Antimony Ores	Mine Drainage	
	Mills	Flotation Process
Titanium Ores	Mine Drainage	
	Mills	
	Mills with Dredge Mining	

* Vanadium extracted from non-radioactive ores

SECTION V

SAMPLING AND ANALYSIS METHODS

The sampling and analysis program discussed in this section was undertaken primarily to implement the Consent Decree and to identify pollutants of concern in the industry, with emphasis on toxic pollutants. A data base has been developed over several years and consists of nine sampling and analysis programs:

1. Screen Sampling Program
2. Verification Sampling Program
3. Verification Monitoring Program
4. EPA Regional Offices Surveillance and Analysis Program
5. Cost Site Visit Program
6. Uranium Study
7. Gold Placer Mining Study
8. Titanium Sand Dredging Mining and Milling Study
9. Solid Waste Study

This section summarizes the purpose of each of these nine sampling efforts, and identifies the sites sampled and parameters analyzed. It also presents an overview of sample collection, preservation, and transportation techniques. Finally, it describes the pollutant parameters quantified, the methods of analyses and the laboratories used, the detectable concentration of each pollutant, and the general approach used to ensure reliability of the analytical data produced. The raw data obtained during these programs are included in Supplement A and are discussed in Section VI, Wastewater Characterization.

SITE SELECTION

The facilities sampled were selected to represent the industry. Considerations included the number of similar operations to be represented; how well each facility represented a subcategory, subdivision, or mill process as indicated by the available data; problems in meeting BPT guidelines or potential problems in meeting BAT guidelines; geographic differences; and treatment processes in use. Successive sampling programs were designed based on data gathered in previous programs.

Several complexes were sampled even though effluent guidelines have not been developed for them. The mine and/or mill waters were sampled separately from the refinery/smelter waters; therefore, data applicable to similar mines and/or mills was developed. Samples taken of refinery and/or smelter water (during the same sampling program) were used to develop a separate guidance document for regulation of complexes.

Screen Sampling Program

Twenty facilities were chosen for initial site visits and sampling to update data previously acquired by EPA and supplied by industry, and to accomplish screen sampling objectives required by the Settlement Agreement.

The Water Quality Control Subcommittee of the American Mining Congress, consisting of representatives from the industry, were presented with descriptions of candidate sites. The comments of the subcommittee were considered and a list of sites to be visited for screen sampling was compiled. At least one facility in each major BPT subcategory was selected. The sites selected were typical of the operations and wastewater characteristics present in particular subcategories.

To determine representative sites, the BPT data base and industry as a whole was reviewed. Consideration was given to:

1. Those using reagents or reagent constituents on the toxic pollutants list.
2. Those using effective treatment for BPT control parameters.
3. Those for which historical data were available as a means of verifying results obtained during screening.
4. Those suspected of producing wastewater streams which contain pollutants not traditionally monitored.

After selection of the facilities to be sampled, a data sheet was developed and sent to each of the 20 operations, together with a letter of notification as to when a visit would be expected. These inquiries led to acquisition of facility information and establishment of industry liaison, necessary for efficient on-site sampling. The information that resulted aided in the selection of the points to be sampled at each site. These sampling points included, but were not limited to, raw and treated effluent streams, process water sources, and intermediate process and/or treatment steps. Copies of the information submitted by each company as well as the contractor's trip report for each visit are contained in the supplements to this report.

Sites visited for screen sampling are listed below by subcategory and facility code:

1. Iron Ore Subcategory - Mine/Mill 1105 and Mine/Mill 1108
2. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory
    - Copper Ore--Mine/Mill 2120, Mine/Mill/Smelter/Refinery 2122, Mine/Mill/Smelter/Refinery 2121, and Mine/Mill/Smelter 2117

- Lead and Zinc Ore--Mine/Mill 3110, Mine/Mill/Smelter/Refinery 3107, and Mine/Mill 3121
- Gold Ore--Mine/Mill 4105
- Silver Ore--Mine/Mill 4401
- Molybdenum Ores--Mill 6101

3. Aluminum Ore Subcategory - Mine 5102
4. Tungsten Ore Subcategory - Mine/Mill 6104
5. Mercury Ore Subcategory - Mill 9202
6. Uranium Ores Subcategory - Mine 9408, Mine 9411, Mine 9402, and Mill 9405
7. Titanium Ore Subcategory - Mine/Mill 9905

These facilities were visited in the period of April through November 1977.

Verification Sampling Program

Sample Set 1. After review of screen sampling analysis results (which are summarized in Section VI), 14 sites were selected for additional sampling visits. Three of these sites were visited to collect additional analytical data on mine/mill/smelter/refinery complexes sampled in screen sampling. Three others, including two not sampled earlier, were visited to collect additional analytical data on treatment systems which were determined to be among the more effective facilities studied during the BPT effort. Because most of the organic toxic pollutants were either not detected or detected only at low concentrations in the screen samples, emphasis was placed on "verification" sampling for total phenol, total cyanide, asbestos (chrysotile), and toxic metals. The following sites were visited in the period of August 1977 through February 1978:

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory
   - Copper Ores--Mine/Mill/Smelter/Refinery 2122 (two trips), Mine/Mill Smelter 2121, and Mine/Mill 2120
   - Lead and Zinc Ores--Mine/Mill/Smelter/Refinery 3107 (two trips), Mine/Mill 3101 (not screen-sampled), and Mine/Mill 3103 (not screen-sampled)

Sample Set 2. Six more facilities, all sampled earlier, were revisited to collect additional data on concentrations of total phenol, total cyanide, and/or asbestos (chrysotile), and to confirm earlier measurements of these parameters. In the period of August 1977 through January 1978 the following sites were visited:

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory only)
   - Silver Ores--Mine/Mill 4401 (asbestos only)
   - Molybdenum Ores--Mill 6101 (asbestos only)

2. Aluminum Ore Subcategory--Mine 5102 (total phenol and asbestos)

3. Uranium Ore Subcategory--Mine 9408 (asbestos only) and Mill 9405 (asbestos only)

Additional Sampling Program

After completion of verification sampling two additional sites were sampled. At the first, a molybdenum mill operation, a complete screen sampling effort was performed to determine the presence of toxic pollutants and to collect data on the performance of a newly installed treatment system. The second facility, a uranium mine/mill, was sampled to collect data on a facility removing radium 226 by ion exchange. Samples collected there were not analyzed for organic toxic pollutants. The following sites were visited in the period of August through November 1978.

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory-(Molybdenum) Mine/Mill 6102

2. Uranium Ore Subcategory - Mine/Mill 9452 (not screen-sampled)

Verification Monitoring Program

Verification monitoring was conducted at three facilities visited during screen sampling to supplement and expand the data for these facilities. The programs lasted from 2 to 12 weeks. Two of the operations were chosen because they had been identified during the BPT study as two of the more treatment efficient facilities. Additional data on long term variations in waste stream characteristics at these sites were needed to supplement the historical discharge monitoring data, and to reflect any recent changes or improvements in the treatment technology used.

The third operation was sampled to determine seasonal variability in the raw and treated waste streams and to supplement existing NPDES monitoring data.

For these monitoring efforts, contractor sampling of the facilities was not economical due to the extended time periods required. Therefore, industry cooperation was solicited, and all monitoring program samples were collected by industry personnel and shipped to the contractor for analysis. Industry personnel were provided detailed instructions on the propoer methods for sample collection, preservation, and transportation. The methods prescribed are the EPA mandated techniques used in the contractor conducted sampling programs and described in the next subsection.

Facilities monitored during the period of September 1977 through March 1978 included:

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory
    - Copper Ores--Mine/Mill/Smelter/Refinery 2122 and Mine/Mill 2120
    - Lead and Zinc Ores--Mine/Mill 3103

Additional information on the monitoring efforts conducted at these facilities is provided in the supplements to this report.

EPA Regional Offices Surveillance and Analysis Program

The data labeled "Surveillance and Analysis" in the supplement to this report was developed by the Agency's regional Sampling and Analysis groups. Fifteen facilities were sampled; ten in the western states of Colorado, Idaho, Wyoming, Montana, and Oregon, one in Arkansas, and four in Missouri. Facilities visited during the period of July through September 1977 were:

1. Copper, Lead, Zinc, Gold Silver, Platinum, and Molybdenum Ore Subcategory
    - Copper Mine/Mill 2120
    - Lead/Zinc Mine/Mill 3107, Mine/Mill 3102, Mine/Mill 3103, Mine/Mill 3109, and Mine/Mill 3119
    - Gold Mill 4102
    - Silver Mills 4401 and 4406, Mine 4402 and Mine/Mill 4403

2. Nickel Ore Subcategory - Mine/Mill/Smelter/Refinery 6106

3. Vanadium Ore Subcategory - Mine/Mill 6107

4. Uranium Ore Subcategory - Mine/Mills 9405 and 9411

Cost Site Visit Sampling

The primary reason for these visits was to determine the cost of implementing particular treatment technologies; therefore, sites were selected by cost considerations. However, many of these sites were sampled previously, and the data obtained from the cost site visits serve to verify the original data.

Facilities visited during the period of September 1979 through January 1980 were:

1. Iron Ore Subcategory - Mine/Mill 1132
2. Copper, Lead, Zinc, Gold, Silver, Platinum, Molybdenum Ore Subcategory
   - Copper Ore--Mine 2110 and Mine/Mill 2116
   - Zinc Ore--Mine/Mill 3106
   - Lead/Zinc Ore--Mine/Mill 3113 and Mine 3117
3. Tungsten Ore--Mine/Mill 6104

Uranium Study

Wastewater sampling was conducted at five uranium mines and at five uranium mills to expand the data base on current state-of-the-art treatment technologies. Uranium mine wastewater treatment technologies studied were barium chloride coprecipitation and ion exchange. Wastewater treatment technologies studied at the five mill sites included barium chloride coprecipitaiton and lime precipitation (for metals removal). The following mine and mill sites were visited during the study:

Uranium Ore Subcategory - Mine 9401, Mine 9402, Mine 9408, Mine 9411, Mine 9412, Mill 9404, Mill 9405, Mill 9414, Mill 9415, Mill 9416.

Gold Placer Mining Study

A sampling effort was conducted to evaluate treatment technologies at Alaskan placer mines. BAT regulations for gold placer mining are reserved for further study. However, several gold placer mining operations, all located in Alaska, were sampled to determine performance capabilities of existing settling ponds used to remove suspended and settleable solids. A summary report has been issued and the data are included in Reference 1. The operations visited as part of this effort are:

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Subcategory - Gold Mines 4126, 4127, 4132, 4133, 4134, 4135, 4136, 4137, 4138, 4143, and 4144.

Titanium Sand Dredging Mining and Milling Study

A study at three titanium dredge mining and milling facilities was conducted to obtain wastewater treatment data on this subcategory of the ore mining and dressing industry. Facilities visited during the period from December 1979 to January 1980 were:

1. Titanium Ore Subcategory - Mine/Mill 9906, Mine/Mill 9907 and Mine/Mill 9910.

The data collected have been summarized in a compehensive report on titanium sand dredging wastewater treatment practices (Reference 2).

Solid Waste Study

The purposes of the solid waste study were to obtain updated wastewater and analytical data on six subcategories and to develop baseline data on the characteristics and amounts of solid waste generated at the facilities selected. One facility in each of the following subcategories was selcted:

1. Aluminum Ore (Mine 5101)
2. Tungsten Ore (Mine/Mill 6105)
3. Nickel Ore (Mine/Mill/Smelter/Refinery 6106)
4. Vanadium Ore (Mine/Mill 6107)
5. Mercury Ore (Mine/Mill 9202)
6. Antimony Ore (Mine/Mill 9901)

SAMPLE COLLECTION, PRESERVATION, AND TRANSPORTATION

Collection, preservation, and transportation of samples were accomplished in accordance with procedures outlined in Appendix III of "Sampling and Analysis Procedures for Screening of Industrial Effluents for Priority Pollutants" (published by the EPA Environmental Monitoring and Support Laboratory, Cincinnati, Ohio, March 1977, revised April 1977) and in "Sampling Screening Procedure for the Measurement of Priority Pollutants" (published by the EPA Environmental Guidelines Division, Washington, D.C., October 1976). The procedures used are summarized in the paragraphs which follow.

In general, four types of samples were collected:

Type 1. A 24-hour composite sample, totaling 9.6 liters (2.5 gallons) in volume, was analyzed for the presence of metals, pesticides and PCBs, asbestos, organic compounds (via gas chromatography/mass spectroscopy (GC/MS) using the liquid/liquid extraction or electron capture methods), and the classical parameters. Usually, this consisted of 200-ml (6.8 ounce) samples, collected and composited at 30-minute intervals by an ISCO Model 1680 peristaltic pump automatic sampler.

When circumstances prevented the use of this sampler, 2.4-liter (81.2 ounce) grab samples were collected and manually composited (also non-flow proportioned) at 6-hour intervals. For example, all tailing samples were composited because the high solids content prevented collection of representative samples with an

ISCO sampler. Also, in the case of one mine/mill/smelter/refinery complex, two consecutive 24-hour composite samples were usually collected to better characterize the several waste streams involved.

Type 2. A 24-hour composite sample, totaling 1 liter (33.8 ounce) in volume, was analyzed for the presence of total cyanide. This was a composite of four 250-ml (8.5 ounce) grab samples collected at 6-hour intervals.

Type 3. A 24-hour composite sample, totaling 0.47 liter (16 ounce) in volume, was analyzed for the presence of phenolics. This was a composite of four 118-ml (4-ounce) grab samples, collected at 6-hour intervals.

Type 4. Two 125-ml (4.2 ounce) grab samples (one a backup sample collected midway in the 24-hour sampling period) were analyzed for the presence of volatile organic compounds by the "purge and trap" method (discussed further under Sample Analysis later in this section).

All sample containers were labeled to indicate sample number, sample site, sampling point, individual collecting the sample, type of sample (e.g., composite or grab, raw discharge or treated effluent), sampling dates and times, preservative used (if any), etc.

Collection and Preservation

Screen, Verification, and Additional Sampling Programs

Whenever practical, all samples collected at each sampling point were taken from mid-channel at mid-depth in a turbulent, well-mixed portion of the waste stream. Periodically, the temperature and pH of each waste stream sampled were measured on-site.

Each large composite (Type 1) sample was collected in a new 11.4-liter (3-gallon), narrow-mouth glass jug that had been washed with detergent and water, rinsed with tap water, rinsed with distilled water, rinsed with methylene chloride, and air dried at room temperature in a dust-free environment.

Before collection of Type 1 samples, new Tygon tubing was cut to minimum lengths and installed on the inlet and outlet (suction and discharge) fittings of the automatic sampler. Two liters (2.1 quart) of blank water, known to be free of organic compounds and brought to the sampling site from the analytical laboratory, were pumped through the sampler and its attached tubing into the glass jug; the water was then distributed to cover the interior of the jug and subsequently discarded.

A blank was produced by pumping an additional 3 liters (3.2 quarts) of blank water through the sampler, distributed inside the glass jug, and poured into a 3.8-liter (1 gallon) sample bottle that had been cleaned in the same manner as the glass jug. The blank sample was sealed with a Teflon-lined cap, labeled, and packed in ice in a plastic foam-insulated chest. This sample subsequently was analyzed to determine any contamination contributed by the automatic sampler.

Metals analyses were run by EPA and Calspan laboratories. During collection of each Type 1 sample, the glass jug was packed in ice in a separate plastic foam-insulated container. After the complete composite sample had been collected, it was mixed to provide a homogeneous mixture, and two 0.95-liter (1-quart) aliquots were removed for metals analysis and placed in labeled, new plastic 0.95-liter bottles which had been rinsed with distilled water. One of these 0.95-liter aliquots was sealed with a Teflon-lined cap, placed in an iced, insulated chest to maintain it at 4 C (39 F), and shipped by air to EPA/Chicago for plasma-arc metal analysis. Initially, the second sample was stabilized by the addition of 5 ml (0.2 ounce) of concentrated nitric acid, capped and iced in the same manner as the first, and shipped by air to the contractor's facility for atomic-absorption metal analysis. The Calspan analyses are reported herein because atomic-adsorption is the preferred technique (except for some beryllium analyses which were taken from plasma-arc data).

Because of subsequent EPA notification that the acid pH of the stabilized sample fell outside the limits permissible under Department of Transportation regulations for air shipment, stabilization of the second sample in the field was discontinued. Instead, this sample was acid-stabilized at the analytical laboratory.

This procedure for obtaining metals samples was not used when the waste streams sampled contained very high concentrations of suspended solids. These solids were generally heavy and rapidly settled out of solution. When samples to be analyzed for metal content were collected from a high solids content stream, they were manually collected and a separate composite sample made (rather than being removed from the 9.6-liter (2.5-gallon) composite). This was necessary to provide a representative sample of the solids fraction.

After removal of the two 0.95-liter (1-quart) metals aliquots from the 9.6-liter (2.5-gallon) composite sample (in the case of low solids content samples), the balance of the sample was sealed in the 11.4-liter (3-gallon) glass jug with a Teflon-lined cap, iced in an insulated chest, and shipped to the Calspan laboratory for further subdivision and analysis for non-volatile organics, asbestos, conventional, and nonconventional parameters. Calspan performed the extraction of organics to be analyzed by GS/MS - liquid/liquid detention and shipped the stable extracts to Gulf South Research. If a portion of this 7.7-liter (2 gallon) sample was requested by an industry representative for independent analysis, a 0.95-liter (1-quart) aliquot was placed in a sample container supplied by the representative.

Sample Types 2 and 3 were stored in new bottles which had been iced and labeled, one liter (33.8-ounce) clear plastic bottles for Type 2, and 0.47-liter (16-ounce) amber glass for Type 3. The bottles had been cleaned by rinsing with distilled water and the samples were preserved as described below.

To each Type 2 (cyanide) sample, sodium hydroxide was added as necessary to elevate the pH to 12 or more (as measured using pH paper). Where the presence of chlorine was suspected, the sample was tested for chlorine (which would decompose most of the cyanide) by using potassium iodide/starch paper. If the paper turned blue, ascorbic acid crystals were slowly added and dissolved until a drop of the sample produced no change in the color of the test paper. An additional 0.6 gram (0.021 ounce) of ascorbic acid was added, and the sample bottle was sealed (by a Teflon-lined cap), labeled, iced and shipped to Calspan for analysis.

To each Type 3 (total phenol) sample, phosphoric acid was added as necessary to reduce the pH to 4 or less (as measured using pH paper). Then, 0.5 gram (0.018 ounce) of copper sulfate was added to kill bacteria, and the sample bottle was sealed (by a Teflon-lined cap), labeled, iced and shipped to Calspan for analysis.

Each Type 4 (volatile organics) sample was stored in a new 125-ml (4.2-ounce) glass bottle that had been rinsed with tap water and distilled water, heated to 105 C (221 F) for 1 hour, and cooled. This method was also used to prepare the septum and lid for each bottle. Each bottle, when used was filled to overflowing, sealed with a Teflon-faced silicone septum (Teflon side down) and a crimped aluminum cap, labeled, and iced. Hermetic sealing was verified by inverting and tapping the sealed container to confirm the absence of air bubbles. (If bubbles were found, the bottle was opened, a few additional drops of sample were added, and a new seal was installed.) Samples were maintained hermetically sealed and iced until analyzed by Gulf South Research.

Verification Monitoring Program

Sampling methods for the monitoring program were similar to those used in the screening and verification efforts. However, the monitoring samples were collected by industry personnel, in contractor-supplied containers and per contractor instructions, over time periods ranging from 2 to 12 weeks. Samples were shipped to Calspan for analysis.

Surveillance and Analysis Program

As discussed previously, the samples for this program were collected and analyzed by Agency regional personnel in three different regions. Techniques were very similar to those described above and EPA approved protocol was observed.

Cost Site Visit Sampling Program

As discussed earlier, this program was primarily designed to collect cost data and sampling was conducted only during the short site visit necessary to gather cost data. Therefore, single grab samples were taken. For total metals and classical pollutant analyses, a one-liter plastic bottle and cap were rinsed several times with the stream to be sampled, and the bottle was filled. For dissolved metals, a portion of this sample was sucked by hand pump through a Millipore 0.45 micron filter into a plastic vacuum flask to rinse the apparatus. This water was discarded, and another quantity filtered, a portion of which was used to rinse a half-liter sample bottle and cap, and the remainder was poured into the rinsed bottle and sealed. The bottles were tightened, and after fifteen minutes tightened again and sealed with plastic tape.

The bottles were stored in a styrofoam chest with ice, and shipped to Radian Corporation.

Sample Transportation

Bottled samples were packed in ice in waterproof plastic foam-insulated chests which were used as shipping containers. Large glass jugs were supported in custom fitted, foam plastic containers before shipping in the insulated chests. All sample shipments were made by air freight.

Associated Data Collection

Drawings and other data relating to plant operations were obtained during site sampling visits. This additional data included detailed information on production, water use, wastewater control, and wastewater treatment practices. Flow diagrams were obtained or prepared to indicate the course of significant wastewater streams. Where possible, control and treatment plant design and cost data were collected, as well as historical data for the sampled waste streams. Information on the use of reagents or products containing chemicals designated as toxic pollutants was also requested.

Uranium Study

Both 24-hour automatic composite samples and grab samples were obtained, depending upon site conditions. It was necessary to collect grab samples at several sites due to freezing conditions; however, company personnel were consulted to ensure that the water quality variations over a 24-hour period were insignificant where grab samples were obtained. The samples were analyzed using EPA methods.

Gold Placer Mining Study

Four-to eight-hour composite samples were collected and shipped to the laboratory for analysis of total suspended solids, mercury and arsenic. All samples analyzed for metal parameters were analyzed for total metal present within the sample. All analyses were performed according to methods described in "Methods for Chemical Analysis of Water and Waste," EPA 600/4-79-020, 1976 and "Standard Methods for the Examination of Water and Wastewater," 1976. Temperature, pH, conductivity, settleable solids, and flow rate measurements were performed on-site using portable instruments.

Titanium Sand Dredging Mining and Milling Study

Sampling and preservation methods employed at two of the facilities studied followed the methods outlined for the screening, verification, and additional sampling programs. The samples were composited over three consecutive, 24-hour periods at designated sites. The samples were analyzed for conventional and toxic pollutants by Radian Corporation using EPA approved methods. Grab samples for conventional and "priority metals" were obtained from a third facility. These samples were also analyzed by Radian Corporation.

Solid Waste Study

The actual waste streams sampled were first identified from the available background data, and then subsequently determined from contact with mine/mill personnel. The number of water and wastewater sample sites chosen at each facility was dependent upon the number of raw waste streams discharging to the treatment system, the number and types of treatment systems utilized, and the known characteristics of the wastewater.

Water and wastewater samples were taken from various locations within the treatment system to obtain the most representative samples from all segments of the system. Due to time and cost limitations, all samples were taken as grab samples. Although the differences between grab and composite samples cannot be fully evaluated here, it is believed that due to the long residence times and large ponds employed in most of the systems studied, the differences between the sampling methods would be slight in most cases.

Wherever possible, the samples were obtained from the middle of the stream in a region of high turbulence to minimize solids separation. In several instances, however, samples were taken from clearwater areas, either because the system had low water levels or was a zero discharge system without recycle. In these cases, samples were obtained near the discharge structure or at a point farthest from the pond influent.

All samples were placed in sample bottles, preserved according to the methods outlined in "Methods for Chemical Analysis of Water and Wastes" EPA-600/4-79-020, and properly labeled. The information on each bottle included the mine/mill name, the sample site, date, and additional sampling information. This information was also recorded in field notes. For samples sent to IFB laboratories, the sample was assigned an EPA sample control number, and the appropriate traffic report was completed. Field data on pH, settleable solids, and temperature were collected to augment the data base.

SAMPLE ANALYSIS

Sampling and Analytical Methods

As Congress recognized in enacting the Clean Water Act of 1977, the state-of-the-art ability to monitor and detect toxic pollutants is limited. Most toxic pollutants were relatively unknown until only a few years ago, and only on rare occasions has EPA regulated or has industry monitored or even developed methods to monitor these pollutants. Section 304(h) of the Act, however, requires the Administrator to promulgate guidelines to establish test procedures for the analysis of toxic pollutants. As a result, EPA scientists, including staff of the Environmental Research Laboratory in Athens, Georgia, and staff of the Environmental Monitoring and Support Laboratory in Cincinnati, Ohio, conducted a literature search and initiated a laboratory program to develop analytical protocols. The analytical techniques used in this study were developed concurrently with the development of general sampling and analytical protocols and were incorporated into the protocols ultimately adopted for the study of other industrial categories. See Sampling and Analysis Procedures for Screening of Industrial Effluents for Priority Pollutants, revised April 1977.

Because Section 304(h) methods were available for most toxic metals, pesticides, cyanide and phenolics (4AAP), the analytical effort focused on developing methods for sampling and analyses of organic toxic pollutants. The three basic analytical approaches considered by EPA are infrared spectroscopy (IS), gas chromatography (GC) with multiple detectors, and gas chromatography/mass spectrometry (GC/MS). Evaluation of these alternatives led the Agency to proposed analytical techniques for 113 toxic organic pollutants (see 44 FR 69464, 3 December 1979, amended 44 FR 75028, 18 December 1979) based on: (1) GC with selected detectors, or high-performance liquid chromatography (HPLC), depending on the particular pollutant and (2) GC/MS. In selecting among these alternatives, EPA considered their sensitivity, laboratory availability, costs, applicability to diverse waste streams from numerous industries, and capability for implementation within the statutory and court-ordered time constraints of EPA's program. The rationale for selecting the proposed analytical protocols may be found in 44 FR 69464 (3 December 1979).

In EPA's judgment, the test procedures used repreesnt the best state-of-the-art methods for toxic pollutant analyses available when this study was begun.

EPA is aware of the continuing evolution of sampling and analytical procedures. Resource constraints, however, prevented the Agency from reworking completed sampling and analysis efforts to keep up with this constant evolution. As state-of-the-art technology progresses, future rulemaking will be initiated to evaluate, and if necessary, incorporate these changes.

Before analyzing ore mining and dressing wastewater, EPA defined specific toxic pollutants for the analyses. The list of 65 pollutants and classes of pollutants potentially includes thousands of specific pollutants, and the expenditure of resources in government and private laboratories would be overwhelming if analyses were attempted for all these pollutants. Therefore, to make the task more manageable, EPA selected 129 specific toxic pollutants for study in this rulemaking and other industry rulemakings. The criteria for selection of these 129 pollutants included frequency of occurrence in water, chemical stability and structure, amount of chemical produced, and availablity of chemical standards for measurement.

As discussed in Sample Collection, EPA collected four types of samples from each sampling point: (1) a 9.6 liter, 24-hour composite sample used to analyzed metals, pesticides, PCBs, asbestos, organic compounds, and the classical parameters; (2) a 1-liter, 24-hour composite sample used to analyze total cyanide; (3) a 0.47-liter, 24-hour composite sample to analyze total phenolics (4AAP); and (4) two 125-ml grab samples to analyze volatile organic compounds by the "purge and trap" method.

EPA analyzed for toxic pollutants according to groups of chemicals and associated analytical schemes. Organic toxic pollutants included volatile (purgeable), base-neutral and acid (extractable) pollutants, and pesticides. Inorganic toxic pollutants included toxic metals, cyanide, and asbestos, (chrysotile and total asbestiform fibers).

The primary method used in screening and verification of the volatile, base-neutral, and acid organics was gas chromatography with confirmation and quantification on all samples by mass spectrometry (GC/MS). Phenolics (total) were analyzed by the 4-aminoantipyrine (4AAP) method. GC was employed for analysis of pesticides with limited MS confirmation. The Agency analyzed the toxic metals by atomic adsorption spectrometry (AAS), with flame or graphite furnace atomization following appropriate digestion of the sample. Samples were analyzed for total cyanide by a colormetric method, with sulfide previously removed by distillation. Asbestos was analyzed by transmission electron microscopy and fiber presence reported as chrysotile and total fiber counts. EPA analyzed for seven other parameters including: pH, temperature, TSS, VSS, COD, TOC, iron, aluminum, and radium 226 (total and dissolved).

The high costs, time-consuming nature of analysis, and limited laboratory capability for toxic pollutant analyses posed considerable difficulties to EPA. The cost of each wastewater analysis for organic toxic pollutants ranges between $650 and $1,700, excluding sampling costs (based on quotations recently obtained from a number of analytical laboratories). Even with unlimited resources, however, time and laboratory capability would have posed additional constraints. Efficiency is improving, but when this study was initiated, a well-trained technician using the most sophisiticated equipment could perform only one complete organic analysis in an eight-hour workday. Moreover, when this rulemaking study began only about 15 commercial laboratories in the United States could perform these analyses. Today, EPA knows of over 50 commercial laboratories that can perform these analyses, and the number is increasing as the demand does.

In plannning data generation for the BAT rulemaking, EPA considered requiring dischargers to monitor and analyze toxic pollutants under Section 308 of the Act. The Agency did not use this authority, however, because it was reluctant to increase the cost to the industry and because it desired to keep direct control over sample analyses in view of the developmental nature of the methodology and the need for close quality control. In addition, EPA believed that the slow pace and limited laboratory capability for toxic pollutant analysis would have hampered mandatory sampling and analysis. Although EPA believes that available data support the BAT regulations, it would have preferred a larger data base for some of the toxic pollutants and will continue to seek additional data. EPA will periodically review these regulations, as required by the Act, and make any revisions supported by new data.

Parameters Analyzed

Analyses varied for the different sampling programs and to simplify the discussion, they are cited by category whenever possible. The categories are:

- Toxics
  - Organics (see Table V-1)
    - All
    - Total Phenolics (4AAP)
  - Metals (see Table V-2)
    - Total
    - Dissolved
  - Cyanide (total)
  - Asbestos
    - Total Fiber

            Chrysotile
    Conventionals
        Total Suspended Solids (TSS)
        pH
    Non-Conventionals
        Temperature
        Volatile Suspended Solids (VSS)
        Chemical Oxygen Demand (COD)
        Total Organic Carbon (TOC)
        Radium 226
            Total
            Dissolved
        Total Phenolics (4AAP)
        Total Settleable Solids
    Miscellaneous Others

The 114 toxic organics (listed in Table V-1), the 13 toxic metals (listed in Table V-2), the conventionals, and the nonconventionals were analyzed in separate groups, with a few exceptions as follows: phenolics, which were occasionally analyzed without the other toxic organics; radium 226, which was only analyzed at uranium facilities; and some exceptions to the toxic metals group, to be noted in the following discussion of specific programs.

Screen Sampling Program

The screen sampling program was designed to build the data base on toxics. Therefore, all mine/mill samples (and most complex samples) were analyzed for the toxics (except dissolved metals). The raw data presented in Supplement 1 include only the parameters detected.

All screen samples were also analyzed for the conventional and nonconventional parameters, with the exception of uranium mines and/or mills, which only analyzed for radium 226.

Verification Sampling Program

Sample Set 1. From the six facilities visited in the screen sampling program, only samples from Mine/Mill/Smelter/Refinery 3107 were screened for toxic organics. Samples from all facilities were analyzed for total metals; two sites were excluded from analysis of cyanide and total phenolics (4AAP). Total fiber and chrysotile asbestos counts were performed on most samples (primarily effluents).

All samples were screened for both conventional parameters, as well as COD and TOC.

Sample Set 2. This set of samples was taken to determine specific parameters, as follows:

Facility	Parameter
Pb/Zn-Mine/Mill 3121	CN
Ag-Mine/Mill 4401	Asbestos
Mo-Mill 6101	Asbestos
Al-Mine 5102	Phenolics, Asbestos
U-Mine 9408	Asbestos
Mill 9405	Asbestos

Additional Sampling Program

Of the two facilities sampled (6102 and 9452), only samples from Mine/Mill 6102 were screened for toxic organics, total fibers, and chrysotile asbestos. No samples from either site were analyzed for cyanide and total phenolics (4AAP).

Samples from both facilities were screened for the conventional pollutants. Mine/Mill 6102 was analyzed for the nonconventional pollutants VSS, COD, and TOC, whereas samples from Mine/Mill 9452 were tested for the nonconventional pollutants COD, chloride, sulfate, uranium, vanadium, and radium 226.

Verification Monitoring Program

Under this program, no analysis of the toxic organics or asbestos was initiated; however, all samples were screened for toxic metals, cyanide and total phenol (4AAP). No sampling analysis of conventional or nonconventional parameters occurred.

Surveillance and Analysis Program

Fourteen facilities were sampled under this program. Samples from all ten were screened for total toxic metals and most samples were tested for total and dissolved toxic metals, cyanide, and total phenolics (4AAP). Of the ten facilities, only samples from Mine/Mill 6107 were analyzed for toxic organics. None of the samples were tested for asbestos.

All samples were screened for conventional pollutants, as well as a wide assortment of nonconventional pollutants, including magnesium, manganese, cobalt, tin, barium, ammonia, settleable solids, and others.

Cost Site Visit Program

The samples were analyzed for all toxic metals (total and dissolved), but not for toxic organics, or cyanide and asbestos.

The samples were screened for both conventionals, and a variety of nonconventionals, such as alkalinity, settleable solids, manganese, and iron.

Uranium Study

Ten uranium mining and milling facilities were sampled during this study. Influent, effluent and intermediate waste streams were analyzed. The wastewater was analyzed for total suspended solids (TSS), cadmium, zinc, arsenic, copper, uranium, molybdenum, vanadium, chemical oxygen demand (COD), pH, sulfate and both total and dissolved radium 226.

Gold Placer Mining Study

Conventional and nonconventional parameters including temperature, pH, conductivity, settleable solids, and total suspended solids were measured. Two toxic metals (arsenic and mercury) were analyzed for total metals present within the sample.

Titanium Sand Dredge Mining and Milling Study

Samples from two facilities were analyzed for toxic organics, toxic metals (total and dissolved), cyanide and asbestos. Toxic metals (total and dissolved), cyanide and asbestos were also measured at a third facility. Conventional parameters and nonconventional parameters such as chemical oxygen demand, total organic carbon, total phenolics (4AAP), iron, manganese, titanium, and oil and grease were also measured.

Solid Waste Program

Wastewater samples were analyzed for conventionals, toxic metals (total and dissolved), and asbestos at each facility sampled. Nonconventionals including chemical oxygen demand, total organic carbon, total phenolics (4AAP), manganese, iron, settleable solids, temperature, and others were measured.

Analytical Methods, Laboratories, and Detection Limits

All parameters were analyzed using methods or techniques described in:

1. "Sampling and Analysis Procedures for Screening of Industrial Effluents for Priority Pollutants" (published by EPA Environmental Monitoring and Support Laboratory, Cincinnati, Ohio, March 1977, revised April 1977).

2. "Analytical Methods for the Verification Phase of the BAT Review (Additional References)" (published by EPA Environmental Guidelines Division, Washington, D.C., June 1977).

3. "Methods for Chemical Analysis of Water and Waste" (published by EPA Environmental Monitoring and Support Laboratory Cincinnati, Ohio, 1974, revised 1976).

4. "Standard Methods for the Examination of Water and Waste Water" (published by the American Public Health

Association, Washington, D.C., 14th edition, 1976).

5. Other EPA approved methods cited in "Guidelines Establishing Test Procedures for the Analysis of Pollutants" (Federal Register, Vol. 41, No. 232, 1 December 1976, pp. 52780-52786).

6. Asbestos analyses were performed using the method outlined in "Preliminary Interim Procedure for Fibrous Asbestos" (published by EPA, Athens, Georgia, undated).

(Note: In accordance with a 13 December 1977 EPA letter, anthracene was added to the sample by Gulf South Research Institute for analysis of organic compounds by GC/MS using the liquid/liquid extraction method.)

The choice of laboratories depended on the analysis to be conducted, and laboratories were changed for different sampling programs.

Detection limits (the lowest concentration at which a parameter can be quantified) vary between sampling programs and even within a program. The detection limit (DL) for a parameter depends on the particular instrument used, the range of standards each set of samples analyzed, the complexity of the sample matrix, and optimization of the instrument response. Therefore, the detection limits given herein are only indicators of the minimum quantifiable concentrations. In fact, some data points are reported below the listed detection limit.

Screen, Verification and Verification Monitoring Programs

The analysis methods, laboratories, and detection limits for samples analyzed during these programs are given in Tables V-3 and V-4. (All of the parameters listed were not analyzed in all four programs.)

Surveillance and Analysis Program

As discussed, this program was conducted by the EPA regional laboratories. The methods used were EPA approved and the detection limits are approximately those shown in Table V-4.

Cost Site Visit Sampling Program

The cost site visit data were generated by Radian Corporation and the methods and detection limits are given in Table V-5.

Uranium, Gold Placer Mining, Titanium Sand Dredging Mining and Milling, and Solid Waste Programs

During these programs many different laboratories were utilized each using EPA approved analytical methods. The detection limits reported are approximately those shown in Table V-4.

Quality Control

Quality control measures used in performing all analyses conducted for this program complied with the guidelines given in "Handbook for Analytical Quality Control in Water and Wastewater Laboratories" (published by EPA Environmental Monitoring and Support Laboratory, Cincinnati, Ohio, 1976). As part of the daily quality control program, blanks (including sealed samples of blank water carried to each sampling site and returned unopened, as well as samples of blank water used in the field), standards, and spiked samples were routinely analyzed with actual samples. As part of the overall program, all analytical instruments (such as balances, spectrophotometers, and recorders) were routinely maintained and calibrated.

The atomic-absorption spectrometer used for metals analysis was checked to see that it was operating correctly and performing within expected limits. Appropriate standards were included after at least every 10 samples. Also, approximately 15 percent of the analyses were spiked with distilled water to assure recovery of the metal of interest. Reagent blanks were analyzed for each metal, and sample values were corrected if necessary.

Total Phenolics (4AAP)

The quality control for total phenolics (4AAP) analysis included demonstrating the quantitative recovery of each phenol distillation apparatus by comparing distilled standards to nondistilled standards. Standards were also distilled each analysis day to confirm the distillation efficiency and purity of reagents. Duplicate and spiked samples were run on at least 15 percent of the samples analyzed for total phenolics.

Cyanide

Similarly, recovery of total cyanide was demonstrated with each distillation-digestion apparatus, and at least one standard was distilled each analysis day to verify distillation efficiency and reagent purity. Quality control limits were established.

During this program, problems were frequently encountered with quality control and analysis of cyanide in mining wastewater samples using the EPA approved Belack Distillation method. Both industry experience and the contractor's laboratory experience indicated problems in obtaining reliable results at the concentrations typically encountered in the metal ore mining industry. Quality control for cyanide included analysis of duplicate samples within a single laboratory and between two laboratories, analysis of spiked samples, and analysis by two different methods. Analysis of duplicated samples often produced results that varied by a factor of 10 (Table V-6). Two commercial laboratories evaluated the analytical recovery of cyanide from mill tailing pond decant samples which had been spiked with cyanide. Samples were delivered to these laboratories immediately after collection to eliminate the possibility of cyanide loss. The results of this quality control program are presented in Table V-7.

A study of the analysis of cyanide in ore mining and processing wastewater was conducted in cooperation with the American Mining Congress to investigate the causes of analytical interferences observed and to determine what effect these interferences had on the precision of the analytical method. Samples of five ore mining and processing wastewaters were obtained along with two municipal wastewater effluents. Cyanide analyses were performed by eight laboratories, including six AMC representatives, EPA's EMSL laboratory in Cincinnati and Radian Corporation's chemical laboratory.

The purpose of this study was to evaluate the EPA-approved method and a modified method for the determination of cyanide. The modified method employed a lead acetate scrubber to remove sulfide compounds produced during the reflux-distillation step. Sulfides have been suspected of providing an interference in the colorimetric determination of cyanide concentrations. Also, several samples were spiked with thiocyanate to ascertain if this compound caused interference in the cyanide analysis.

A statistical analysis of the resultant data shows no significant difference in precision or accuracy of the two methods employed when applied to metal ore mining and milling wastewaters having cyanide concentrations in the 0.2 mg/l to 0.4 mg/l range. Based upon the statistical analysis, approximately 50 percent of the overall error of either method was attributed to intralaboratory error. This highlights the need for an experienced analyst to perform cyanide analyses.

Initial cyanide concentrations were determined by the EPA approved cyanide analysis method. Samples which were found to contain less than 0.2 mg/l cyanide were spiked with sodium cyanide and potassium ferricyanide. Samples containing over 0.5 mg/l cyanide were air sparged at pH 2 for 24 hours and filtered through 0.45u membrane filters prior to raising the pH above 10. The following table summarizes the initial and adjusted cyanide concentrations.

Sample	Initial CN by Approved Method (mg/l)	Theoretical CN After Adjustment (mg/l)
A	0.26	0.26 (no adjustment)
B	0.02	0.210
C	0.02	2.73
D	0.54*	0.54 (no adjustment)
E	0.02	0.389
F	84.00	2.5 - 3.5**
G	0.02	0.273

*Sample showed strong interferences during colorimetric procedure.

**Results were not repeatable.

Sample D contained approximately 180 mg/l of thiocyanate. Samples B and E were spiked with 33 mg/l and 100 mg/l of thiocyanate, respectively. Sodium thiocyanate was used as the thiocyanate source.

The results of this study are summarized in Table V-8. Based upon these data, the following conclusions have been drawn for ore mining and processing wastestreams containing 0.2 to 0.4 mg/l cyanide:

1. The overall relative standard deviation for the EPA-approved method was 27.6 percent.
2. The overall relative standard deviation for the modified method was 30.4 percent.
3. Accuracy as average percent deviation from the standards was -12.6 for the approved method and -6.1 for the modified method. Neither of these values is significantly different from zero for this sample size.
4. The approved and modified methods work equally well for the analysis of cyanide in ore mining and processing wastewaters.
5. No major problems were demonstrated in the cyanide analysis by either method for samples containing 30 to 100 mg/l of thiocyanate.

Based upon the relative standard deviations calculated, it can be said that for an ore mining or processing wastewater sample containing 0.2 mg/l of cyanide, 95 percent of the analyses would be between 0.08 and 0.32 mg/l using the modified method and between 0.09 and 0.31 mg/l using the approved method. Over 99 percent of the analyses would be between 0.02 and 0.38 mg/l using the modified method and between 0.035 and 0.365 mg/l using the approved method (Reference 3). Accordingly, the Agency must allow for analytical measurement of up to .4 mg/l for total cyanide. (See discussion Section X).

PCBs and Pesticides

In analyses of pesticides and PCBs, extraction efficiencies were determined. Known standards were prepared, extracted, and analyzed to guarantee minimum extraction/concentration losses. Calibrations of all analytical components were carried out with high-quality pure materials. A calibration mixture was run daily to ensure that retention time and instrumentation response for each parameter analyzed did not change due to column and detector aging.

Table V-1

TOXIC ORGANICS

Compound Name

1. *acenaphthene (B)***
2. *acrolein (V)***
3. *acrylonitrile (V)
4. *benzene (V)
5. *benzidene (B)
6. *carbon tetrachloride (tetrachloromethane) (V)

*Chlorinated benzenes (other than dichlorobenzenes)

7. chlorobenzene (V)
8. 1,2,4-trichlorobenzene (B)
9. hexachlorobenzene (B)

*Chlorinated ethanes(including 1,2-dichloroethane, 1,1,1-trichloroethane and hexachloroethane)

10. 1,2-dichloroethane (V)
11. 1,1,1-trichlorethane (V)
12. hexachlorethane (B)
13. 1,1-dichloroethane (V)
14. 1,1,2-trichloroethane (V)
15. 1,1,2,2-tetrachloroethane (V)
16. chloroethane (V)

*Chloroalkyl ethers (chloromethyl, chloroethyl and mixed ethers)

17. bis (chloromethyl) ether (B)
18. bis (2-chloroethyly) ether (B)
19. 2-chloroethyl vinyl ether (mixed) (V)

*Chlorinated naphthalene

20. 2-chloronaphthalene (B)

*Chlorinated phenols (other than those listed elsewhere; includes trichlorophenols and chlorinated cresols)

21. 2,4,6-trichlorophenol (A)***
22. parachlorometa cresol (A)
23. *chloroform (trichloromethane) (V)
24. *2-chlorophenol (A)

Table V-1 (Continued)

TOXIC ORGANICS

*Dichlorobenzenes

25. 1,2-dichlorobenzene (B)
26. 1,3-dichlorobenzene (B)
27. 1,4-dichlorobenzene (B)

*Dichlorobenzidine

28. 3,3'-dichlorobenzidine (B)

*Dichloroethylenes (1,1-dichloroethylene and 1,2-dichloroethylene)

29. 1,1-dichloroethylene (V)
30. 1,2-trans-dischloroethylene (V)
31. *2,4-dichlorophenol (A)

*Dichloropropane and dichloropropene

32. 1,2-dichloropropane (V)
33. 1,2-dichloropropylene (1,3-dichloropropene) (V)
34. *2,4-dimenthylphenol (A)

*Dinitrotoluene

35. 2,4-dinitrotoluene (B)
36. 2,6,-dinitrotoluene (B)
37. *1,2-diphenylhydrazine (B)
38. *ethylbenzene (V)
39. *fluoranthene (B)

*Haloethers (other than those listed elsewhere)

40. 4-chlorophenyl phenyl ether (B)
41. 4-bromophnyl phenyl ether (B)
42. bis(2-chloroisopropyl) ether (B)
43. bis(2-chloroethoxy) methane (B)

*Halomethanes (other than those listed elsewhere)

44. methylene chloride (dichloromethane) (V)
45. methyl chloride (chloromethane) (V)
46. methyl bromide (bromomethane) (V)
47. bromoform (tribromomethane) (V)
48. dichlorobromomethane (V)

Table V-1 (Continued)

TOXIC ORGANICS

49. trichlorofluoromethane (V)
50. dichlorodifluoromethane (V)
51. chlorodibromomethane (V)
52. *hexachlorobutadiene (B)
53. *hexachlorocyclopentadiene (B)
54. *isophorone (B)
55. *naphthalene (B)
56. *nitrobenzene (B)

*Nitrophenols (including 2,4-dinitrophenol and dinitrocesol)

57. 2-nitrophenol (A)
58. 4-nitrophenol (A)
59. *2,4-dinitrophenol (A)
60. 4,6-dinitro-o-cresol (A)

*Nitrosamines

61. N-nitrosodimethylamine (B)
62. N-nitrosodiphenylamine (B)
63. N-nitrosodi-n-propylamine (B)
64. *pentachlorophenol (A)
65. *phenol (A)

*Phthalate esters

66. bis(2-ethylhexyl) phthalate (B)
67. butyl benzyl phthalate (B)
68. di-n-butyl phthalate (B)
69. di-n-octyl phthalate (B)
70. diethyl phthalate (B)
71. dimethyl phthalate (B)

*Polynuclear aromatic hydrocarbons

72. benzo (a)anthracene (1,2-benzanthracene) (B)
73. benzo (a)pyrene (3,4-benzopyrene) (B)
74. 3,4-benzofluoranthene (B)
75. benzo(k)fluoranthane (11,12-benzofluoranthene) (B)
76. chrysene (B)
77. acenaphthylene (B)
78. anthracene (B)
79. benzo(ghi)perylene (1,12-benzoperylene) (B)
80. fluorene (B)
81. phenathrene (B)

Table V-1 (Continued)

TOXIC ORGANICS

82. dibenzo (a,h)anthracene (1,2,5,6-dibenzanthracene) (B)
83. indeno (1,2,3-cd)(2,3,-o-phenylenepyrene) (B)
84. pyrene (B)
85. *tetrachloroethylene (V)
86. *toluene (V)
87. *trichloroethylene (V)
88. *vinyl chloride (chloroethylene) (V)

Pesticides and Metabolites

89. *aldrin (P)
90. *dieldrin (P)
91. *chlordane (technical mixture and metabolites) (P)

*DDT and metabolites

92. 4,4'-DDT (P)
93. 4,4'-DDE(p,p'DDX) (P)
94. 4,4'-DDD(p,p'TDE) (P)

*endosulfan and metabolites

95. a-endosulfan-Alpha (P)
96. b-endosulfan-Beta (P)
97. endosulfan sulfate (P)

*endrin and metabolites

98. endrin (P)
99. endrin aldehyde (P)

*heptachlor and metabolites

100. heptachlor (P)
101. heptachlor epoxide (P)

*hexachlorocyclohexane (all isomers)

102. a-BHC-Alpha (P) (B)
103. b-BHC-Beta (P) (V)
104. r-BHC (lindane)-Gamma (P)
105. g-BHC-Delta (P)

Table V-1 (Continued)

TOXIC ORGANICS

*polychlorinated biphenyls (PCB's)

106. PCB-1242 (Arochlor 1242) (P)
107. PCB-1254 (Arochlor 1254) (P)
108. PCB-1221 (Arochlor 1221) (P)
109. PCB-1232 (Arochlor 1232) (P)
110. PCB-1248 (Arochlor 1248) (P)
111. PCB-1260 (Arochlor 1260) (P)
112. PCB-1016 (Arochlor 1016) (P)

113. *Toxaphene (P)
114. **2,3,7,8-tetrachlorodibenzo-p-dioxin (TCDD)

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*Specific compounds and chemical classes as listed in the consent degree.
**This compound was specifically listed in the consent degree.
***B = analyzed in the base-neutral extraction fraction
V = analyzed in the volatile organic fraction
A = analyzed in the acid extraction fraction

Table V-2

TOXIC METALS, CYANIDE AND ASBESTOS

1. *Antimony (Total)
2. *Arsenic (Total)
3. *Asbestos (Fibrous)
4. *Beryllium (Total)
5. *Cadmium (Total)
6. *Chromium (Total)
7. *Copper (Total)
8. *Cyanide (Total)
9. *Lead (Total)
10. *Mercury (Total)
11. *Nickel (Total)
12. *Selenium (Total)
13. *Silver (Total)
14. *Thallium (Total)
15. *Zinc (Total)

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*Specific compounds and chemical classes as listed in the consent degree.

TABLE V-3. POLLUTANTS ANALYZED AND ANALYSIS TECHNIQUES/LABORATORIES

SUBSTANCE	ANALYSIS TECHNIQUE/LAB.	
pH	SM	Calspan
total suspended solids	SM	↓
volatile suspended solids	SM	↓
COD	SM	↓
TOC	SM	↓
radium 226 (total)	RA	Teledyne Isotopes
radium 226 (dissolved)	RA	Teledyne Isotopes
antimony (total)	AA	Calspan
arsenic (total)	AA	Calspan
beryllium (total)	AA, PA	Calspan + EPA
cadmium (total)	AA	Calspan
chromium (total)	↓	↓
copper (total)	↓	↓
lead (total)	↓	↓
mercury (total)	↓	↓
nickel (total)	↓	↓
selenium (total)	↓	↓
silver (total)	↓	↓
thallium (total)	↓	↓
zinc (total)	↓	↓
asbestos (fibrous)	TEM	↓
cyanide (total)	SM	↓
phenol (total)	SM	↓
aldrin	EC	↓
dieldrin	↓	↓
chlordane (technical mixture and metabolites)	↓	↓
4,4'-DDT	↓	↓
4,4'-DDE (p,p'-DDX)	↓	↓
4,4'-DDD (p,p'-TDE)	↓	↓
α-endosulfan	↓	↓
β-endosulfan	↓	↓
endosulfan sulfate	↓	↓
endrin	↓	↓
endrin aldehyde	↓	↓
heptachlor	↓	↓
heptachlor epoxide	↓	↓
α-BHC	↓	↓
β-BHC	↓	↓
γ-BHC (lindane)	↓	↓
δ-BHC	↓	↓
PCB-1242 (Arochlor 1242)	↓	↓
PCB-1254 (Arochlor 1254)	↓	↓
acenaphthene	LL	Gulf South Research Inst.
acrolein	PT	↓
acrylonitrile	PT	↓
benzene	PT	↓
benzidine	LL	↓
carbon tetrachloride (tetrachloromethane)	PT	↓
chlorobenzene	PT	↓
1,2,4 trichlorobenzene	LL	↓
hexachlorobenzene	LL	↓
1,2-dichloroethane	PT	↓
1,1,1-trichloroethane	PT	↓
hexachloroethane	LL	↓
1,1-dichloroethane	PT	↓
1,1,2 trichloroethane	PT	↓
1,1,2,2 tetrachloroethane	PT	↓
chloroethane	PT	↓
bis (chloromethyl) ether	LL, PT	↓
bis (chloroethyl) ether	LL	↓
2-chloroethyl vinyl ether (mixed)	PT	↓
2-chloronaphthalene	LL	↓
2,4,6 trichlorophenol	LL	↓
parachlorometa cresol	LL	↓
chloroform (trichloromethane)	PT	↓
2-chlorophenol	LL	↓
1,2-dichlorobenzene	LL	↓

SUBSTANCE	ANALYSIS TECHNIQUE/LAB.	
1,3-dichlorobenzene	LL	Gulf South Research Inst.
1,4-dichlorobenzene	LL	↓
3,3'-dichlorobenzidine	LL	↓
1,1-dichloroethylene	PT	↓
1,2-trans-dichloroethylene	PT	↓
2,4-dichlorophenol	LL	↓
1,2-dichloropropane.	PT	↓
1,3-dichloropropylene (1,3-dichloropropene)	PT	↓
2,4-dimethylphenol	LL	↓
2,4-dinitrotoluene	LL	↓
2,6-dinitrotoluene	LL	↓
1,2-diphenylhydrazine	LL	↓
ethylbenzene	PT	↓
fluoranthene	LL	↓
4-chlorophenyl phenyl ether	LL	↓
4-bromophenyl phenyl ether	LL	↓
bis (2-chloroisopropyl) ether	LL	↓
bis (2-chloroethoxy) methane	LL	↓
methylene chloride (dichloromethane)	PT	↓
methyl chloride (chloromethane)	↓	↓
methyl bromide (bromomethane)	↓	↓
bromoform (tribromomethane)	↓	↓
dichlorobromomethane	↓	↓
trichlorofluoromethane	↓	↓
dichlorodifluororomethane	↓	↓
chlorodibromomethane	↓	↓
hexachlorobutadiene	LL	↓
hexachlorocyclopentadiene	↓	↓
isophorone	↓	↓
napthalene	↓	↓
nitrobenzene	↓	↓
2-nitrophenol	↓	↓
4-nitrophenol	↓	↓
2,4-dinitrophenol	↓	↓
4,6-dinitro-o-cresol	↓	↓
N-nitrosodimethylamine	↓	↓
N-nitrosodiphenylamine	↓	↓
N-nitrosodi-n-propylamine	↓	↓
pentachlorophenol	↓	↓
phenol	↓	↓
bis (2-ethylhexyl) phthalate	↓	↓
butyl benzyl phthalate	↓	↓
di-n-butyl phthalate	↓	↓
diethyl phthalate	↓	↓
dimethyl phthalate	↓	↓
1,2 benzanthracene	↓	↓
benzo (a) pyrene (3,4-benzopyrene)	↓	↓
3,4- benzofluorathene	↓	↓
11,12 benzofluoranthene	↓	↓
chrysene	↓	↓
acenaphthylene	↓	↓
anthracene	↓	↓
1,12 benzoperylene	↓	↓
fluorene	↓	↓
phenanthrene	↓	↓
1,2 5,6-dibenzanthracene	↓	↓
indeno (1,2,3-c,d) pyrene	↓	↓
pyrene	↓	↓
2,3,7,8 tetrachlorodibenzo-p-dioxin (TCDD)	↓	↓
tetrachlorethylene	PT	↓
toluene	PT	↓
trichloroethylene	PT	↓
vinyl chloride	PT	↓
toxaphene	LL	↓

AA = Atomic Absorption spectroscopy (flame or flameless)
EC = Gas Chromatography by Electron-Capture detection method
LL = Gas Chromatography/Mass Spectrometry (GC-MS) by Liquid Liquid Extraction method
PA = Emission Spectroscopy using inductively coupled argon plasma (Plasma Arc method)
PT = Gas Chromatography/Mass Spectrometry (GC-MS) by Purge-and-Trap method
RA = Radioassay using scintillation counter or proportional counter
TEM = Transmission Electron Microscopy
SM = Other standard (EPA-approved) methods

TABLE V-4. LIMITS OF DETECTION FOR POLLUTANTS ANALYZED

SUBSTANCE	CONCENTRATION
pH	–
total suspended solids	1 mg/l
volatile suspended solids	1 mg/l
COD	2 mg/l
TOC	1 mg/l
radium 226 (total)	1 pCi/l
radium 226 (dissolved)	1 pCi/l
antimony (total)	0.2 mg/l
arsenic (total)	0.002 mg/l *
beryllium (total)	0.005 mg/l
cadmium (total)	0.002 mg/l
chromium (total)	0.02 mg/l
copper (total)	0.01 mg/l
lead (total)	0.05 mg/l
mercury (total)	0.0005 mg/l **
nickel (total)	0.02 mg/l
selenium (total)	0.002 mg/l *
silver (total)	0.01 mg/l
thallium (total)	0.1 mg/l
zinc (total)	0.005 mg/l
asbestos (fibrous) (total)	2.2 x 10^5 fibers/l
cyanide (total)	0.02 mg/l
phenol (total)	0.002 mg/l
aldrin	0.1 μg/l
dieldrin	0.5 μg/l
chlordane (technical mixture and metabolites)	1 μg/l
4,4'-DDT	1 μg/l
4,4'-DDE (p,p'-DDX)	1 μg/l
4,4'-DDD (p,p'-TDE)	1 μg/l
α-endosulfan	1 μg/l
β-endosulfan	1 μg/l
endosulfan sulfate	1 μg/l
endrin	0.5 μg/l
endrin aldehyde	0.5 μg/l
heptachlor	0.1 μg/l
heptachlor epoxide	0.1 μg/l
α-BHC	0.1 μg/l
β-BHC	0.1 μg/l
γ-BHC (lindane)	0.1 μg/l
δ-BHC	0.1 μg/l
PCB-1242 (Arochlor 1242)	1 μg/l
PCB-1254 (Arochlor 1254)	1 μg/l
acenaphthene	0.03 μg/l
acrolein	no estimate
acrylonitrile	no estimate
benzene	0.04 μg/l
benzidine	~ 25.0 μg/l
carbon tetrachloride (tetrachloromethane)	0.35 μg/l
chlorobenzene	0.15 μg/l
1,2,4-trichlorobenzene	~ 1.0 μg/l
hexachlorobenzene	~ 1.0 μg/l
1,2-dichloroethane	0.08 μg/l
1,1,1-trichloroethane	0.15 μg/l
hexachloroethane	~ 0.10 μg/l
1,1-dichloroethane	0.15 μg/l
1,1,2-trichloroethane	0.10 μg/l
1,1,2,2-tetrachloroethane	0.90 μg/l
chloroethane	~ 0.50 μg/l
bis (chloromethyl) ether	no estimate
bis (chloroethyl) ether	no estimate
2-chloroethyl vinyl ether (mixed)	~ 0.50 μg/l
2-chloronaphthalene	0.06 μg/l
2,4,6-trichlorophenol	~ 1.0 μg/l
parachlorometa cresol	~ 1.0 μg/l
chloroform (trichloromethane)	0.05 μg/l
2-chlorophenol	1.00 μg/l
1,2-dichlorobenzene	0.10-0.20 μg/l
1,3-dichlorobenzene	0.10-0.20 μg/l
1,4-dichlorobenzene	0.10-0.20 μg/l
3,3'-dichlorobenzidine	~ 25.0 μg/l
1,1-dichloroethylene	0.50 μg/l
1,2-trans-dichloroethylene	0.35 μg/l
2,4-dichlorophenol	~ 1.0 μg/l
1,2-dichloropropane	0.02 μg/l
1,3-dichloropropylene (1,3-dichloropropene)	0.35 μg/l
2,4-dimethylphenol	0.40 μg/l
2,4-dinitrotoluene	0.20 μg/l
2,6-dinitrotoluene	0.75 μg/l
1,2-diphenylhydrazine	0.20 μg/l
ethylbenzene	0.10 μg/l
fluoranthene	0.20 μg/l
4-chlorophenyl phenyl ether	0.06 μg/l
4-bromophenyl phenyl ether	~ 0.80 μg/l
bis (2-chloroisopropyl) ether	no estimate
bis (2-chloroethoxy) methane	no estimate
methylene chloride (dichloromethane)	0.08 μg/l
methyl chloride (chloromethane)	0.35 μg/l
methyl bromide (bromomethane)	0.08 μg/l
bromoform (tribromomethane)	0.40 μg/l
dichlorobromomethane	0.05 μg/l
trichlorofluoromethane	~ 0.10 μg/l
dichlorodifluororomethane	~ 0.10 μg/l
chlorodibromomethane	~ 0.10 μg/l
hexachlorobutadiene	0.08 μg/l
hexachlorocyclopentadiene	no estimate
isophorone	0.10 μg/l
napthalene	0.15 μg/l
nitrobenzene	0.85 μg/l
2-nitrophenol	1.0 μg/l
4-nitrophenol	no estimate
2,4-dinitrophenol	will not chromatograph
4,6-dinitro-o-cresol	will not chromatograph
N-nitrosodimethylamine	not established
N-nitrosodiphenylamine	not established
N-nitrosodi-n-propylamine	not established
pentachlorophenol	~ 50.0 μg/l
phenol	1.0 μg/l
bis (2-ethylhexyl) phthalate	0.20 μg/l
butyl benzyl phthalate	0.25 μg/l
di-n-butyl phthalate	0.3-0.4 μg/l
diethyl phthalate	0.20 μg/l
dimethyl phthalate	0.35 μg/l
1,2-benzanthracene	0.05 μg/l
benzo (a) pyrene (3,4-benzopyrene)	~ 0.5 μg/l
3,4- benzofluorathene	~ 0.5 μg/l
11,12-benzofluoranthene	~ 0.5 μg/l
chrysene	0.20 μg/l
acenaphthylene	~ 0.5 μg/l
anthracene	0.05 μg/l
1,12-benzoperylene	~ 0.5 μg/l
fluorene	0.10 μg/l
phenanthrene	0.05 μg/l
1,2,5,6-dibenzanthracene	~ 0.5 μg/l
indeno (1,2,3-c,d) pyrene	~ 0.5 μg/l
pyrene	0.40 μg/l
2,3,7,8-tetrachlorodibenzo-p-dioxin (TCDD)	no estimate
tetrachloroethylene	1.10 μg/l
toluene	0.35 μg/l
trichloroethylene	0.35 μg/l
vinyl chloride	~ 0.1 μg/l
toxaphene	not established

*As analyzed by atomic-absorption spectroscopy using gaseous hydride (a flame method)

**As analyzed by atomic-absorption spectroscopy using cold vapor technique (a flameless method)

Table V-5

LIMITS OF DETECTION FOR POLLUTANTS ANALYZED FOR COST - SITE VISITS BY RADIAN CORPORATION

Parameter	Method of Analyses	Detection Limit (ppm)
Sb	AAS* - hydride generation	0.005
As	AAS - hydride generation	0.002
Be	AAS - flameless - HGA**	0.001
Cd	ICPES***	0.005
Cr	ICPES	0.005
Cu	ICPES	0.005
Fe	ICPES	0.004
Pb	AAS - flameless - HGA	0.002
Hg	AAS - flameless - cold vapor	0.001
Mn	ICPES	0.001
Ni	ICPES	0.020
Se	AAS - hydride generation	0.005
Ag	ICPES	0.005
Tl	AAS - flameless - HGA	0.002
Zn	ICPES	0.002

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*Atomic Absorption Spectrophotometry
**Inductively Coupled Argon Plasma Emission Spectrometry
***Heated Graphite Analyzer

TABLE V-6. COMPARISON OF SPLIT SAMPLE ANALYSIS* FOR CYANIDE BY TWO DIFFERENT LABORATORIES USING THE BELACK DISTILLATION/PYRIDINE-PYROZOLONE METHOD

Sample Description	Analytical Result (mg Total CN/l)	
	Lab #1	Lab #2
1. Tailing Pond Influent	0.02	0.06
2. Tailing Pond Influent	0.05	<0.02
3. Tailing Pond Influent	0.08	0.03
4. Tailing Pond Effluent	0.04	0.50
5. Tailing Pond Effluent	0.04	0.47
6. Tailing Pond Effluent	0.03	0.57

*All samples collected at the same time at Mine/Mill 3121

TABLE V-7. ANALYTICAL QUALITY CONTROL PERFORMANCE OF COMMERCIAL LABORATORY PERFORMING CYANIDE* ANALYSES BY EPA APPROVED BELACK DISTILLATION METHOD

SAMPLE DESCRIPTION	CYANIDE AS CN mg/l	% ANALYTICAL RECOVERY OF SPIKE
Distilled H_2O + 0.1 mg/l CN as NaCN	0.125	125
Distilled H_2O + 0.2 mg/l CN as NaCN	0.250	125
Distilled H_2O + 0.4 mg/l CN as NaCN	0.200	50
Tailing Pond Decant (no spike)	0.069	–
Tailing Pond Decant Spiked with 0.1 mg/l CN as NaCN	0.144	85
Tailing Pond Decant Spiked with 0.2 mg/l CN as NaCN	0.119	44
Tailing Pond Decant Spiked with 0.4 mg/l CN as NaCN	0.136	29
Tailing Pond Decant Spiked with 1.0 mg/l CN as $K_3Fe(CN)_6$	0.028 0.032	3 3
Tailing Pond Decant Spiked with 1.0 mg/l CN as $K_4Fe(CN)_6$	0.027 0.079	3 7

*All cyanide analyses are total cyanide

Table V-8

SUMMARY OF CYANIDE ANALYSIS DATA FOR SAMPLES OF ORE MINING AND PROCESSING WASTEWATERS

Sample	Method*	Number of Labs	Theoretical CN concentration mg/1	Mean Recovery mg/1	Standard Deviation mg/1	Intralaboratory Standard Deviation mg/1
B	M	7	0.210	0.176	0.080	0.042
	A	3		0.139	0.059	
E	M	7	0.389	0.353	0.117	
	A	3		0.338	0.064	0.025
C	M	7	2.73	3.04	1.15	1.87
	A	3		2.72	0.89	
F	M	7	**	2.66	1.23	
	A	3		3.27	0.283	0.431
A	M	7	0.26	0.268	0.062	0.049
	A	3		0.233	0.073	
G	M	7	0.273	0.290	0.030	
	A	3		0.285	0.018	0.053
D	M	7	***	0.269	0.123	
	A	2		0.175	0.108	

*A = EPA Approved Cyanide Analysis Method
M = Modified Cyanide Analysis Method
**Results were not repeatable in the initial analysis
***Strong interferences were noted in the initial analysis

SECTION VI

WASTEWATER CHARACTERIZATION

The data base developed during the sampling program described in Section V is presented in Supplement A and summary tables are presented and discussed in this section. Also, a summary of reagent usage at flotation mills, the largest users of mill process chemicals, is presented to evaluate mill reagents as potential sources of toxic pollutants. Special circumstances; such as, the presence of certain toxic pollutants in mine water as a result of backfilling mines with mill tailings, are discussed at the end of this section.

SAMPLING PROGRAM RESULTS

The analytical results of the nine sampling programs discussed in Section V are presented in Supplement A and were entered into a computerized data base. Using this data base, summary data tables were generated for the entire category; and each subcategory, subdivision, and mill process (Tables VI-1 through VI-18, which may be found at the end of this section). These tables include raw and treated wastewater data; and the range of pollutant concentrations observed is indicated by the mean and median values, and the 90 percent and maximum values (defined below).

All Subcategories Combined

Table VI-1 summarizes the BAT data base for all the mines and mills in all subcategories in the ore mining and dressing point source category. As indicated by the table, only 27 of the toxic organics were detected in the category's treated wastewater. Organic compounds are not found naturally with metal ores. Introduction of organics during froth flotation mill processing is discussed later in this section. Otherwise, the discussion of toxic organics is left to Section VII, Selection of Pollutant Parameters.

Toxic metals are naturally associated with metal ores and all of the 13 toxic metals were found in wastewater from the category. The concentrations of each metal varied greatly, as expected for such a diverse category. Cyanide and asbestos, also toxic parameters, were observed in many samples and in varied concentrations.

The conventional parameters observed were primarily those regulated by BPT effluent guidelines, that is TSS and pH. The TSS values are very high in many raw samples because tailings samples which typically run in the tens of thousands of mg TSS/l are included in "raw" samples. Effluent TSS values vary, but are generally low indicating good solids settling characteristics.

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Values of pH vary, but are often in the alkaline range (7 to 14). This is because several mill processes operate at elevated pHs. As indicated by discussions in Section III, pH, TSS, and metals values are closely allied. The solubility of many metals varies greatly with pH, and the status of the metals (dissolved v. solubilized) affects the concentration of TSS. This relationship is used by the industry for ore beneficiation and for wastewater treatment.

Nonconventional parameters such as COD, TOC, volatile suspended solids (VSS), and iron were also analyzed for many samples. The concentrations of the organic related parameters, COD, TOC, and VSS, were always low. Any organic compounds added in mill processes are not indicated by these tests which are designed to measure relatively large masses of organics (in the mg/l range at a minimum). Iron is common in metal ores and the summary table reflects this.

The entire BAT data set is discussed below by subcategory, subdivision, and as a mill process or mine drainage, and these discussions more completely characterize mine/mill wastewaters. In general it can be noted from Table VI-1 that organic compounds are not the major concern in this category (a point discussed thoroughly in Section III), metals are prevalent, pH values are generally alkaline, and cyanide and asbestos are often present.

Iron Subcategory, Mine Drainage Subdivision

Table VI-2 summarizes the data for iron mines. Many of the toxic metals were not detected in the one or two available samples; arsenic (.005 mg/l) copper (.090 and 120 mg/l), and zinc (.018 and .030 mg/l) are the exceptions. Asbestos fibers, both total and chrysotile, were detected in relatively small amounts compared to the rest of the category (see Table VI-1). Generally, (comparing Tables VI-1 and VI-2) iron mine water is characterized by low pollutant levels. This is true of most mine water and is the reason for separate mine and mill subdivisions.

Iron Subcategory, Mill Subdivision, Physical and/or Chemical Mill Processes

As indicated in Table VI-3, several of the toxic metals were present in the one or two raw samples taken, but most are removed by existing treatment technologies (sedimentation) and were not detected in discharge samples. Copper is the least affected by current treatment methods. Asbestos was detected in relatively high concentrations in the raw sample (compared to Table VI-1), and in lower concentrations in the discharged sample. This indicates that current treatment methods are removing a portion of the asbestos; a conclusion supported by Table VI-3. The COD, VSS, and TOC (indicators of gross organic pollution) are somewhat higher than the rest of the industry (compared to Table VI-1), but they are effectively removed by current technologies. Iron was detected in one raw sample, as expected for iron mills, but was below detection in the discharge water. Several toxic metals, asbestos, TSS, and some nonconventional parameters were found in the raw wastewater of iron mills, but these parameters were reduced during treatment and many do not appear in the discharge water.

Copper/Lead/Zinc/Gold/Silver/Platinum/Molybdenum Subcategory, Mine Drainage Subdivision

This subcategory includes more mines than any other subcategory and more samples are available for characterization than for other subcategories. As shown in Table VI-4, all of the toxic metals were detected at least four times in sixteen raw samples. High median concentrations (relative to the other metals detected) of antimony, arsenic, cadmium, chromium, copper, lead, nickel, thallium, and zinc are shown in Table VI-4 for raw mine drainage. In the discharged water, however, the metals concentrations are lower, with the median values ranging from not detected to 280 ug/l (zinc).

Cyanide, asbestos, and phenolics are other toxic parameters detected in this subdivision. Cyanide is used in the froth flotation process and backfilling mines with mill tailings can cause cyanide to pollute the mine water. Asbestos, being a mineral, is found with many metal ores, although the concentrations reported in Table VI-4 are relatively low (compared to Table VI-1) and have a small range for samples taken at many types of mines. Phenolics were detected at low concentrations.

Copper/Lead/Zinc/Silver/Gold/Platinum/Molybdenum Subcategory, Cyanidation Mill Process

This subdivision was regulated as no discharge of process wastewater in BPT effluent guidelines, therefore, few samples were taken in BAT sampling programs and no discharge samples were taken. It can be seen from Table VI-5 that many toxic parameters, including cyanide, were found in high concentrations in this mill water; thereby supporting the BPT no discharge requirement.

Copper/Lead/Zinc/Silver/Gold/Platinum/Molybdenum Subcategory, Mill Subdivision, Froth Flotation Mill Process

There were more samples of this mill process than of any others because froth flotation is a widely used process with the potential to generate wastewater polluted with many toxics. As seen in Table VI-6, all of the toxic metals were detected in raw mill water. The number of detections ranged from 7 to 78 out of 78 samples and median concentrations ranged from 1.1 ug/l (mercury) to 63,300 ug/l (copper). These wide ranges are due to the variations in the ore milled at different locations. Generally, the metals concentrations are in the high range of values reported for the category as a whole (Table VI-1). The discharged concentrations of metals are, generally, one or two orders of magnitude lower than the raw values. The number of toxic metals with median concentration over 20 ug/l are reduced from ten in raw samples to five in treated samples and, overall, the concentrations are reduced by existing treatment.

Asbestos, cyanide, and phenolics were also detected in both raw and discharged samples. Median values for all were above the respective medians for the whole category (Table VI-1). All were reduced by the existing treatment systems.

Nonconventional parameters and TSS were generally high (compared to Table VI-1) and the pH range is great.

Generally, mill water and tailings from this mill process contain a wider range and higher concentrations of pollutants, including toxics, than other mill processes or mines in this category. The various process reagents used in flotation are discussed later in this section.

Copper/Lead/Zinc/Gold/Silver/Platinum/Molybdenum Subcategory, Mill Subdivision, Heap/Vat/Dump/In-Situ Leaching

Very few samples were taken in this mill process because it is regulated as no discharge of process water in BPT effluent guidelines. As can be seen in Table VI-7, the raw wastewater has high concentration of several parameters, the reason for the no discharge requirement. The one discharged sample reported is actually treated recycle water which is not discharged.

Copper/Lead/Zinc/Gold/Silver/Platinum/Molybdenum Subcategory, Placer Operations Recovering Gold

A study was conducted in 1978 to evaluate current wastewater handling practices at gold placer mines. Eleven operations, all located in Alaska, were sampled to determine performance capabilities of existing settling ponds. Only two of the toxic metals were monitored during the program, arsenic and mercury. Settleable solids were also monitored to provide an indication of treatment pond performance. As can be seen in Table VI-8, the settleable solids concentrations range from not detected to 500 ml/l/hr. However, many of the different samples are discharges that had not been treated in settling ponds.

Aluminum Subcategory, Mine Drainage Subdivision

As shown in Table VI-9, aluminum mine drainage is low in most pollutants. The toxic metals present in the discharge are in relatively low concentrations (compared to Table VI-1) and are chromium, copper, mercury, nickel, and zinc. Asbestos was present in moderate concentrations (compared to Table VI-1) and was not affected by the existing treatment methods. Acid pH levels were noted in the raw, but these increased to the alkaline range (7 pH 14) after pH adjustment.

Tungsten Subcategory, Mill Subdivision

As shown in Table VI-10, 13 of the toxic metals were detected in the raw wastewater. However, these are reduced during treatment leaving only seven above 20 ug/l in the discharge. Of these, copper, lead, and zinc have high concentrations (compared to the other discharge metals concentrations).

Asbestos and phenolics were detected in the raw samples; cyanide was not. The values of asbestos are high relative to the category as a whole (see Table VI-1). The effluent phenolics are low relative to the values in Table VI-1.

Mercury Subcategory, Mill Subdivision

As seen in Table VI-11, the toxic metals are found in high concentrations in the raw wastewater in this subdivision, as are asbestos and phenolics. That is why the applicable BPT regulation is no discharge of process wastewater. The discharged sample in Table VI-11 is actually treated recycle water.

Uranium Subcategory, Mine Drainage Subdivision

Uranium mine drainage, is, relative to mill water less polluted. As seen in Table VI-12, many of the toxic metals were detected, all but zinc in concentrations less than 65 ug/l. Only six were detected in the treated samples, none greater than 50 ug/l.

Cyanide was not detected, and phenolics were detected at a low concentration (10 ug/l). Asbestos was detected in both raw and treated samples at moderate concentrations (as compared to Table VI-1).

Not listed in Table VI-12, but shown in the support data (Supplement A), are radium 226 concentrations. Uranium ore is radioactive and radium 226 is a radionuclide always associated with uranium. It is one of the uranium decay series and has a half life of 1,620 years. Raw mine water may have several hundred to a thousand pico-Curies per liter (p Ci/l) of Ra 226, but existing treatment is capable of reducing this to the BPT guideline of 10 p Ci/l (total, 30-day average).

Uranium Subcategory, Mill Subdivision

As seen in Table VI-13, several of the toxic metals are found in both raw and treated wastewater. Treated wastewater in this table is actually recycle water. The facilities do not discharge. This recycle water is not treated specifically for metals, and, therefore, little reduction occurs.

Asbestos was found in both influent and effluent samples in moderate concentrations (as compared to Table VI-1). Cyanide was not detected and total phenol (4AAP) were detected at a low concentration (10 ug/l). As with mine drainage, mill water may have several hundred to a thousand p Ci/l Ra 226. Current treatment at the single uranium mill discharging is reducing this to 10 p Ci/l, the BPT limitation.

Titanium Subcategory, Mine Subdivision

As can be seen in Table VI-14, the mine water from this subcategory is relatively clean (relative to Table VI-1). Three toxic metals (copper, lead, and zinc) were detected at 20 ug/l. Relative to the category as a whole (Table VI-1), the asbestos values are low. Total phenolics were detected at 30 ug/l.

Titanium Subcategory, Mill Subdivision

As shown in Supplement A (Support Data; Sample Points 1A and 2A, for Mill 9905), seven toxic metals were detected in the raw wastewater; all but selenium and lead at concentrations greater than 200 ug/l. These concentrations were reduced by treatment, leaving only five detected toxic metals ranging in concentration from 20 to 100 ug/l.

Asbestos was detected at moderate concentrations (compared to Table VI-1). Cyanide was not detected and phenolics were detected at 10 ug/l in raw and discharged samples.

Titanium Subcategory, Mills with Dredge Mining Subdivision

Table VI-15 summarizes the data for the titanium mills employing dredge mining. Ten toxic metals were detected in the raw water, at concentrations less than or equal to 80 ug/l. In the treated effluent, six toxic metals were detected. Only zinc was detected in concentrations greater than 10 ug/l.

COD and TOC concentrations in the raw water were generally present in higher concentrations than the rest of the category due to the presence of organic material in some of the ores. The treatment processes used substantially reduced the concentrations of both COD and TOC. The TSS concentration of the effluents were less than 10 mg/l.

Vanadium Subcategory, Mine Drainage Subdivision

Table VI-16 illustrates the character of vanadium mine drainage. Several toxic metals were present both in the raw and discharged water. Discharge concentrations greater than 20 ug/l were reported for chromium, copper, lead, nickel, and zinc. Cyanide and total phenolics were not detected. The asbestos values were low relative to the category as a whole.

Vanadium Subcategory, Mill Subdivision

As seen in Table VI-17, many toxic metals were detected in both the raw and discharged waters from this subdivision. Of the metals, only mercury was reduced below the detection limit by the existing treatment system. Cyanide was also reduced below the detection limit, and no total phenolics were detected in raw or discharged water.

Antimony Subcategory, Mill Subdivision

The data for this subcategory are presented in Table VI-18. There is no discharge of treated wastewater from the single mill in this subdivision. Relatively high concentrations of antimony and arsenic are present in the raw and treated wastewater. Phenolics were not detected in the raw or treated wastewater. Asbestos was detected in moderate concentrations compared to Table VI-1. The pH of the impounded water was greater than 12.0.

REAGENT USE IN FLOTATION MILLS

Froth flotation processes use various reagents in the porcess, and these reagents are discharged with the tailings and mill process water. Flotation reagents are a possible source of toxic organics in an industry which, otherwise, has no known source of toxic organics. Therefore, a survey was conducted to determine the availability of toxic organics and other toxics in flotation reagents.

The results of a nationwide survey of sulfide ore flotation mills indicate that over 547,400 metric tons (602,000 short tons) of chemical flotation reagents were consumed in 1975 (Reference 1). Reagent use data supplied by 22 milling operations indicate that 63 different chemical compounds are used directly in sulfide ore flotation circuits. These reagents are categorized as:

1. pH Modifier (Conditioner, Regulator)--Any substance used to regulate or modify the pH of an ore pulp or flotation process stream. Examples of the most commonly used reagents are lime, soda ash (sodium carbonate), caustic soda (sodium hydroxide), and sulfuric acid.
2. Promoter (Collector)--A reagent added to a pulp stream to bring about adherence between solid particles and air bubbles in a flotation cell. Examples of the most common promoters are xanthate and dithiophosphate salts, as well as saturated hydrocarbons (such as fuel oil).
3. Frother--A substance used in flotation processing to stabilizeair bubbles, principally by reducing surface tension. Common frothers are pine oil, cresylic acid, amyl alcohol, MIBC, and polyglycol methyl ethers.

4. Activator--A substance which, when added to a mineral pulp, promotes flotation in the presence of a collecting agent. It may be used to increase the floatability of a mineral in a froth or to refloat a depressed mineral. A good example of an activating agent is copper sulfate, used in the flotation of sphalerite.

5. Depressant--A substance which reacts with the particle surface to render it less prone to stay in the froth, thus causing it to wet down as a tailing product (contrary to activator). Examples of depressing agents most commonly used are cyanide, zinc sulfate, corn starch, sulfur dioxide, and sodium sulfite.

Table VI-19 summarizes reagent use for copper, lead, zinc, silver, and molybdenum flotation mills which discharge process wastewater. Comparing the reagents listed in Table VI-19 to the list of toxic pollutants given in Section V, only the following reagents are considered to be potential sources of one or more toxic pollutants in mill process wastewater: copper, zinc, chromium, and total phenolics (4AAP).

Copper

Copper sulfate addition to a flotation pulp containing sphalerite (ZnS) is a good example of an activating agent. The cupric ions replace zinc in the sphalerite lattice to permit better collector attachment, thus allowing the mineral to be floated with a xanthate (Reference 2). Copper ammonium chloride functions in much the same manner and is used at one operation (Mill 3110) because it is purchased as a waste byproduct from the manufacturer of electronic circuit boards. Copper sulfate is highly soluble in water and is added to the flotation circuit in concentrations as high as 100 mg/l (as Cu). Residual dissolved copper in the tailings pulp stream readily forms copper hydroxide precipitates at the alkaline pH common to most sulfide flotation systems.

Zinc

The function of zinc sulfate is the depression of sphalerite when floating galena and copper sulfides (Reference 3), and the mechanism involved is very similar to that of copper sulfate described above. Typically, dosage rates of 0.1 to 0.4 kilogram of zinc sulfate per metric ton (0.2 to 0.8 pound per short ton) of ore feed are used, often in conjunction with cyanide. These dosage rates translate to dissolved zinc loads in the flotation circuit of 5.2 to 65 mg/l (as Zn). Residual zinc concentrations from excessive zinc sulfate use are small compared to the total zinc content of the tailings.

Chromium

Sodium dichromate is used as a flotation reagent at only one of the 22 flotation mills listed in Table VI-19. It functions as a depressant for galena in copper/lead separations. Dosages of this reagent are relatively small, and long term analyses of treated effluent have not indicated the presence of chromium in detectable concentrations.

Cyanide

Sodium cyanide and, to a lesser extent, calcium cyanide have found widespread application within the industry as strong depressants for iron sulfides and sphalerite. Cyanide also acts as a mild depressant for chalcopyrite, enargite, bornite, and most other sulfide minerals with the exception of galena (Reference 4). A secondary action of cyanide, in some instances, may be the cleaning of tarnished mineral surfaces, thereby allowing a more selective separation of the individual minerals (Reference 5). Typical cyanide reagent dosages range from 0.003 to 0.125 kilogram per metric ton (0.006 to 0.250 pound per short ton) of ore feed and average 0.029 (0.058). Expressed in terms of water use, cyanide dosages range from less than 1.0 to 50.4 milligrams per liter (as sodium cyanide), with an average of about 11.

Sodium cyanide and calcium cyanide flotation reagents are the sole source of cyanide in flotation mill effluents. Four flotation mills (2122, 3121, 6101, and 6102) have effluent discharge concentrations of 0.1 mg/l total cyanide or greater. Mill 6102 is the largest consumer of cyanide in terms of dosage per unit of ore feed and per unit of flotation circuit water feed. As a result, Mill 6102 produces a raw discharge with total cyanide concentrations of 0.2 to 0.4 mg/l. Cyanide dosages used at Mills 2122, 3121, and 6101 are consistent with amounts used throughout the industry, and, for this reason, reagent use alone does not appear to be the cause for high cyanide levels. The treatment of cyanide-bearing wastewater and the chemistry of cyanide in mill wastewater are discussed in Section VIII of this report.

Phenolic Compounds

"Reco" (sodium dicresyldithiophosphate) is used at Mill 2122 to promote the flotation of copper sulfide minerals. Reco is similar to American Cyanamid's AEROFLOAT 31 and 242 promoters, which are used at Mills 3101, 3104, 3115, 4403, and 9202. These reagents contain the cresyl group ($CH_3.C_6H_3.OH$), a very close relative of the toxic substance 2,4-dimethylphenol, which has been detected in raw mill wastewater samples collected during the toxic substance screen sampling program at Mills 2122 and 9202. Mills 3101, 3104, 3115 and 4403 were not selected as sites for screen and/or verification sampling of organic toxic pollutants during this program.

Cresylic acid is used as a flotation reagent at Mills 2117, 2121, and 4403. Xylenols, $C_2H_5.C_6H_4OH$ or $(CH_3)_2.C_6H_3.OH$, are the dominant constituents of commercial cresylic acids and include the toxic pollutant, 2,4-dimethyphenol, which has not been detected in raw or treated wastewater samples at Mills 2117 and 2121. Mill 4403 was not sampled for the organic toxic substances. Nitrobenzenes are present in Aero 633, but nitrobenzene was not detected in wastewater during this program. However, screening and verification sample data strongly implicate these phenol-based flotation reagents as the sources of total phenol (4AAP) in mill process wastewaters. From a practical standpoint, cresylic acid can be considered as 100 percent phenolic with the relative phenolic content of the other phenol-containing reagents being considerably less. Phenolic concentrations of 5.2 mg/l and 5.0 mg/l have been detected in the mill tailing samples at Mill 2117, and treated effluent samples were found to contain 0.30 mg/l and 0.36 mg/l on 2 consecutive days. The large consumption of cresylic acid at Mill 2117 (0.035 kilogram/metric ton equivalent to 0.070 pound per short ton, of ore) and the consistency of data substantiate cresylic acid as being a significant source of phenolic compounds in flotation mill process effluents.

Phenolic compounds were found to be the most prevalent toxic organic species detected in the screen samples, but concentrations did not exceed 0.03 mg/l except at operations which are known to employ one or more of the phenol based flotation reagents previously discussed.

SPECIAL PROBLEM AREAS

Backfilling of Mines With Mill Sand Tailings

A review of sample data and historical monitoring data supplied by the industry indicates the presence of significant concentrations of cyanide in several mine water discharges. Further examination revealed that the facilities with cyanide in mine water backfilled mined-out stopes using mill sand tailings from flotation circuits which use cyanide compounds as process reagents.

A variety of undergound mining techniques are used throughout the mining industry. Typical mining methods include room-and-pillar, vein (or drift) mining, open stoping, pillar stoping, cut-and-fill, and panel-and-fill. The selection of method(s) is dependent on many factors, such as the type and shape of the ore deposit, the depth of excavations, and the ground conditions.

Cut-and-fill, pillar stoping, and panel-and-fill techniques have found common application in lead, zinc, and silver mines located in Colorado, Utah, and the Coeur d'Alene Mining District in Idaho. An inherent feature of these mining methods is the refilling of worked-out and abandoned stopes and other workings to prevent subsidence and cave-ins as mining progresses through the ore body. For many years, waste rock from the mine exploration crosscuts was used as fill material; however, the development of hydraulic sandfill procedures has simplified the backfill operation. In current practice, the coarse (sand) fraction of the flotation-mill tailings is often segregated from the tailings pulp stream by hydro-cyclones and pumped into the mine for backfilling.

Nine mines (Mines 3107, 3113, 3120, 3121, 3130, 4104, 4105, 4401 and 4402) are known to practice hydraulic backfilling with mill sand tailings. Eight of these nine mills use cyanide either as a flotation reagent (Mills 3107, 3113, 3121, 3130 and 4401) or as a leaching agent (Mills 4104, 4105, and 4402). The nature of the mechanism by which cyanide depresses pyrite and sphalerite is such that much of the cyanide added to the flotation circuit associates with the depressed minerals in the tailings and ultimately is leached into mine water during hydraulic backfill.

Mine 3130 is the only facility with a separate mine drainage treatment system that periodically monitors for cyanide. Effluent monitoring data (summarized in Section VIII) include cyanide analyses of five 24-hour composite samples collected during the period of June 1977 through October 1977. The data indicate that cyanide concentrations in the treated mine water did not exceed 0.2 mg/l total cyanide for mills and mine/mills on a daily basis, although the monthly average exceeded 0.1 mg/l on one occasion. Examination of raw (untreated) mine-water data from Mine 3130 indicates that cyanide is not effectively removed by the treatment system, which consists of lime and flocculant addition, followed by a series of two sedimentation ponds. This treatment is not designed for destruction or removal of cyanide and, does not provide sufficient residence time for natural aeration. fore, the poor removals observed are not surprising.

Total cyanide concentrations detected in five mine-water grab samples collected to support BAT at Mine 3130 were found to range from 0.04 to 0.16 mg/l. A 24-hour composite mine-water sample collected at Mine 3107 was found to contain 0.4 mg/l during backfill operations.

Mine 4105, located in South Dakota, was visited during the screening phase of this program. Analysis of mine water for total cyanide indicated that, for the days when the contractor sampled, concentrations were less than detectable. During previous visits to this facility, no cyanide was detected in mine water samples.

Table VI -1

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
ACENAPHTHENE	33	0					*	28	0				
ACROLEIN	33	0					*	28	0				
ACRYLONITRILE	33	0					*	28	0				
BENZENE	33	10	4.8922	4	10	10	*	28	3	8.3333	7	10.7	11
BENZIDENE	33	0					*	28	0				
CARBON TETRACHLORIDE	33	1	1	1	1	1	*	28	0				
CHLOROBENZENE	33	0					*	28	1	0.005	0.005	0.005	0.005
1,2,3-TRICHLOROBENZENE	33	0					*	28	0				
HEXACHLOROBENZENE	33	0					*	28	0				
1,2-DICHLOROETHANE	33	0					*	28	0				
1,1,1-TRICHLOROETHANE	33	9	6.7208	6.5811	10	10	*	28	5	7.2649	6.5811	10	10
HEXACHLOROETHANE	33	0					*	28	0				
1,1-DICHLOROETHANE	33	0					*	28	0				
1,1,2-TRICHLOROETHANE	33	0					*	28	0				
1,1,2,2-TETRACHLOROETHAN	33	0					*	28	0				
CHLOROETHANE	33	0					*	28	0				
BIS(CHLOROMETHYL) ETHER	33	0					*	28	0				
BIS(2-CHLOROETHYL) ETHER	33	0					*	28	0				
2-CHLOROETHYL VINYL ETHE	32	0					*	28	0				
2-CHLORONAPHTHALENE	32	0					*	28	0				
2,4,6-TRICHLOROPHENOL	32	1	11.667	11.667	11.667	11.667	*	28	0				
PARACHLOROMETA CRESOL	32	0					*	28	0				
CHLOROFORM	32	9	7.6096	3.1623	12.5	35	*	28	8	5.1281	3.1623	10	10
2-CHLOROPHENOL	32	0					*	28	0				
1,2-DICHLOROBENZENE	32	0					*	28	0				
1,3-DICHLOROBENZENE	32	0					*	28	0				
1,4-DICHLOROBENZENE	32	0					*	28	0				
3,3-DICHLOROBENZIDINE	32	0					*	28	0				

Table VI -1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	80%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	80%	MAX
1,1-DICHLOROETHYLENE	32	2	6.5811	3.1623	8.6325	10	*	28	0				
1,2-TRANS-DICHLOROETHYLE	32	0					*	28	1	270	270	270	270
2,4-DICHLOROPHENOL	32	1	10	10	10	10	*	28	0				
1,2-DICHLOROPROPANE	32	0					*	28	0				
1,3-DICHLOROPROPENE	32	0					*	28	0				
2,4-DIMETHYLPHENOL	32	1	140	140	140	140	*	28	1	270	270	270	270
2,4-DINITROTOLUENE	32	0					*	28	0				
2,6-DINITROTOLUENE	32	0					*	28	0				
1,2-DIPHENYLHYDRAZINE	32	0					*	28	0				
ETHYLBENZENE	32	4	6.7167	1	13.48	17.667	*	28	3	6.6	4.9	9.64	10
FLUORANTHENE	32	0					*	28	0				
METHYL CHLORIDE	33	1	45	45	45	45	*	28	0				
METHYL BROMIDE	33	0					*	28	0				
BROMOFORM	33	0					*	28	0				
DICHLOROBROMOMETHANE	33	0					*	28	2	6.5811	3.1623	8.6325	10
TRICHLOROFLUOROMETHANE	33	5	5.0325	2.0811	10	10	*	28	3	4.7208	2.0811	7.9487	10
DICHLORODIFLUOROMETHANE	33	0					*	28	0				
CHLORODIBROMOMETHANE	33	0					*	28	0				
HEXACHLOROBUTADIENE	33	0					*	28	0				
HEXACHLOROCYCLOPENTADIEN	33	0					*	28	0				
ISOPHORONE	33	0					*	28	0				
NAPHTHALENE	33	1	12.5	12.5	12.5	12.5	*	28	0				
NITROBENZENE	33	0					*	28	0				
2-NITROPHENOL	33	0					*	28	0				
4-NITROPHENOL	33	0					*	28	0				
2,4-DINITROPHENOL	33	0					*	28	0				
4,6-DINITRO-O-CRESOL	33	0					*	28	0				

Table VI -1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
N-NITROSODIMETHYLAMINE	33	0					*	28	0				
N-NITROSODIPHENYLAMINE	33	0					*	28	0				
N-NITROSODI-N-PROPYLAMIN	33	0					*	28	0				
PENTACHLOROPHENOL	33	1	10	10	10	10	*	28	0				
PHENOL	33	2	118	78	143.2	160	*	28	3	92.3	33.45	168.8	210
BIS(2-ETHYLHEXYL) PHTHAL	33	15	20.16	13	39.833	100	*	28	18	12.458	10	28	50
BUTYL BENZYL PHTHALATE	33	2	10.75	0.5	18.9	21	*	28	4	27.791	10	52.4	68
DI-N-BUTYL PHTHALATE	33	13	16.489	10	26.1	56	*	28	12	25.864	10	39.2	140
DI-N-OCTYL PHTHALATE	10	3	10	10	10	10	*	7	3	12.167	10	14.56	16.5
DIETHYL PHTHALATE	33	16	24.414	10	59.4	90	*	28	4	7.875	9.8	10	10
DIMETHYL PHTHALATE	33	0					*	28	3	12.2	5.8	20.35	25
BENZO(A)ANTHRACENE	33	0					*	28	0				
BENZO(A)PYRENE	33	0					*	28	0				
BENZO(B)FLUORANTHENE	33	0					*	28	0				
BENZO(K)FLUORANTHENE	33	0					*	28	0				
CHRYSENE	33	0					*	28	0				
ACENAPHTHYLENE	33	0					*	28	0				
ANTHRACENE	33	0					*	28	0				
BENZO(G,H,I)PERYLENE	33	0					*	26	0				
FLUORENE	33	1	10	10	10	10	*	28	1	10	10	10	10
PHENANTHRENE	33	0					*	28	0				
DIBENZO(A,H)ANTHRACENE	33	0					*	28	0				
INDENO(1,2,3-C,D)PYRENE	33	0					*	28	0				
PYRENE	33	0					*	28	0				
TETRACHLOROETHYLENE	33	2	7.75	4.5	9.7	11	*	28	1	1.1	1.1	1.1	1.1
TOLUENE	33	9	399.28	2.0811	368.3	3580	*	28	6	2.5967	1	6.28	10
TRICHLOROETHYLENE	33	0					*	28	0				

Table VI -1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
VINYL CHLORIDE	33	0					*	28	0				
ALDRIN	33	4	6.4156	5	9	10	*	28	2	6.5811	3.1623	8.6325	10
DIELDRIN	33	0					*	28	2	6.5811	3.1623	8.6325	10
CHLORDANE	33	0					*	28	0				
4,4-DDT	33	0					*	28	0				
4,4-DDE	33	1	5	5	5	5	*	28	0				
4,4-DDD	33	1	6.6667	6.6667	6.6667	6.6667	*	28	0				
ENDOSULFAN-ALPHA	33	1	10	10	10	10	*	28	0				
ENDOSULFAN-BETA	33	0					*	28	0				
ENDOSULFAN SULFATE	33	0					*	28	0				
ENDRIN	33	0					*	28	1	5	5	5	5
ENDRIN ALDEHYDE	33	0					*	28	0				
HEPTACHLOR	33	1	7.5	7.5	7.5	7.5	*	28	2	6.5811	3.1623	8.6325	10
HEPTACHLOR EPOXIDE	33	0					*	28	0				
BHC-ALPHA	33	5	5.2849	4.0811	7.5	10	*	28	3	5	5	5	5
BHC-BETA	33	5	6.1325	5	8.75	10	*	28	1	5	5	5	5
BHC (LINDANE)-GAMMA	33	4	6.2072	5	6.6667	10	*	28	0				
BHC-DELTA	33	2	5	5	5	5	*	28	2	5	5	5	5
PCB-1242 (AROCHLOR 1242)	33	0					*	28	0				
PCB-1254 (AROCHLOR 1254)	33	0					*	28	0				
PCB-1221 (AROCHLOR 1221)	9	0					*	6	0				
PCB-1232 (AROCHLOR 1232)	9	0					*	6	0				
PCB-1248 (AROCHLOR 1248)	8	0					*	6	0				
PCB-1260 (AROCHLOR 1260)	9	0					*	6	0				
PCB-1016 (AROCHLOR 1016)	8	0					*	6	0				
TOXAPHENE	32	0					*	27	0				
2,3,7,8-TETRACHLORODIBEN	33	0					*	28	0				

Table VI -1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)							TREATED (UG/L)					
			DETECTED VALUES ONLY							DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MED	90%	MAX
CIS 1-3-DICHLOROPROPYLEN							*						
TRAN 1,3-DICHLOROPROPYLE							*						

Table VI-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(MG/L)							TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	DETECTED VALUES ONLY MEDIAN	90%	MAX		NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	DETECTED VALUES ONLY MEDIAN	90%	MAX
ANTIMONY (TOTAL)	82	6	.04233	0.028	0.1	0.1	*	71	3	0.034	100E-5	0.1	0.1
ARSENIC (TOTAL)	114	166	2.2219	0.5	8.43	12	*	100	83	.15349	0.018	0.6	1.5
BERYLLIUM (TOTAL)	94	43	.13735	0.1	0.2782	0.92	*	73	10	0.0051	0.005	0.0109	0.011
CADMIUM (TOTAL)	196	54	0.213	0.0225	0.655	1.2	*	92	36	.01415	0.005	0.06	0.077
CHROMIUM (TOTAL)	85	70	3.8162	0.81	11	18	*	75	26	.13623	0.035	0.332	1.8
COPPER (TOTAL)	163	160	66.221	3.8	224.5	464	*	90	13	.23964	0.06	0.548	4.6
CYANIDE (TOTAL)	66	24	.32842	0.21	0.93	1.24	*	57	14	.13571	0.08	0.435	8.6
LEAD (TOTAL)	66	70	4.1807	1.35	4.98	130	*	75	31	.13481	0.05	0.376	0.959
MERCURY (TOTAL)	87	54	.00399	.00145	0.014	0.02	*	80	37	.01264	800E-6	0.0306	0.25
NICKEL (TOTAL)	86	70	3.6166	2.5	9.4	14.2	*	75	43	.22202	0.07	0.966	1.28
SELENIUM (TOTAL)	84	56	0.2506	0.15	0.615	1.5	*	73	37	.05957	0.015	0.112	0.9
SILVER (TOTAL)	84	24	.34524	0.21	0.77	1.1	*	73	9	.01567	0.019	0.04	0.04
THALLIUM (TOTAL)	82	3	.04433	1.17	1.24	1.29	*	71	3	.52767	0.74	0.84	0.84
ZINC (TOTAL)	196	196	37.885	6.875	202.4	300	*	92	82	.90209	0.062	2.244	11.1
COD	22	22	302.72	10.95	1761	1900	*	23	23	11.698	11	20.6	53
TSS	47	47	3156.8	40	836	95430	*	47	46	24.989	9.5	69.2	157
TOC	6	6	485	560	750	750	*	9	8	5.1875	5.23	6	6
PH (UNITS)	35	35	6.9934	7	8.44	9.5	*	56	56	6.9714	7.7	8.16	8.5
PHENOLICS (4AAP)	72	71	.11372	0.04	0.3422	0.75	*	57	49	.07378	0.032	0.21	0.46
IRON (TOTAL)	22	19	523.7	6.248	1900	2040	*	25	21	.62619	0.209	2.192	3.87

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Table VI-2

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY IRON
SUBDIVISION MINE
MILL PROCESS MINE DRAINAGE

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	2	0					*	1	0				
ARSENIC (TOTAL)	1	0					*	1	1	0.005	0.005	0.005	0.005
BERYLLIUM (TOTAL)	2	0					*	1	0				
CADMIUM (TOTAL)	2	0					*	1	0				
CHROMIUM (TOTAL)	1	0					*	1	0				
COPPER (TOTAL)	2	1	0.09	0.09	0.09	0.09	*	1	1	0.12	0.12	0.12	0.12
CYANIDE (TOTAL)	1	0					*	1	0				
LEAD (TOTAL)	2	0					*	1	0				
MERCURY (TOTAL)	2	0					*	1	0				
NICKEL (TOTAL)	2	0					*	1	0				
SELENIUM (TOTAL)	2	0					*	1	0				
SILVER (TOTAL)	2	0					*	1	0				
THALLIUM (TOTAL)	2	0					*	1	0				
ZINC (TOTAL)	2	1	0.018	0.018	0.018	0.018	*	1	1	0.03	0.03	0.03	0.03
COD	1	1	10	10	10	10	*	1	1	6	6	6	6
VSS	1	1	2	2	2	2	*	1	1	3	3	3	3
TSS	2	2	4.65	4.65	5	5	*	1	1	4	4	4	4
TOC	1	1	25	25	25	25	*	1	1	19	19	19	19
PH (UNITS)	2	2	8.075	8.075	8.15	8.15	*	1	1	8	8	8	8
PHENOLICS (4AAP)	1	0					*	1	0				
IRON (TOTAL)	1	0					*						
ASBESTOS (CHRYSO) (F/L)	1	1	3500E3	3500E3	3500E3	3500E3	*	1	1	3800E3	3800E3	3800E3	3800E3
TOTAL FIBERS (F/L)	1	1	1700E4	1700E4	1700E4	1700E4	*	1	1	4200E4	4200E4	4200E4	4200E4

Table VI-3

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY IRON
SUBDIVISION MILL
MILL PROCESS PHYSICAL AND/OR CHEMICAL

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	2	0					*	2	0				
ARSENIC (TOTAL)	2	1	0.89	0.89	0.89	0.89	*	2	1	0.005	0.005	0.005	0.005
BERYLLIUM (TOTAL)	2	1	0.92	0.92	0.92	0.92	*	2	0				
CADMIUM (TOTAL)	2	1	0.031	0.031	0.031	0.031	*	2	0				
CHROMIUM (TOTAL)	2	2	0.276	0.276	0.5	0.5	*	2	2	0.014	0.014	0.018	0.018
COPPER (TOTAL)	2	2	0.225	0.225	0.32	0.32	*	2	1	0.1	0.1	0.1	0.1
CYANIDE (TOTAL)	1	0					*	1	0				
LEAD (TOTAL)	2	2	0.0505	0.0505	0.08	0.08	*	2	0				
MERCURY (TOTAL)	2	0					*	2	0				
NICKEL (TOTAL)	2	2	2.15	2.15	2.7	2.7	*	2	0				
SELENIUM (TOTAL)	2	1	0.02	0.02	0.02	0.02	*	2	0				
SILVER (TOTAL)	2	2	0.017	0.017	0.02	0.02	*	2	0				
THALLIUM (TOTAL)	2	0					*	2	0				
ZINC (TOTAL)	2	2	3.15	3.15	5.8	5.8	*	2	2	0.019	0.019	0.03	0.03
COD	1	1	96	96	96	96	*	1	1	4	4	4	4
VSS	1	1	80	80	80	80	*	1	1	.03162	.03162	.03162	.03162
TSS	2	2	64500	64500	110000	110000	*	2	2	2.0158	2.0158	4	4
TOC	1	1	22	22	22	22	*	1	1	11	11	11	11
PH (UNITS)	2	2	7.775	7.775	7.9	7.9	*	2	2	7.675	7.675	8.1	8.1
PHENOLICS (4AAP)	1	0					*	1	0				
IRON (TOTAL)	1	1	73	73	73	73	*	1	0				
ASBESTOS (CHRYSO) (F/L)	1	1	3800E7	3800E7	3800E7	3800E7	*	1	1	4100E3	4100E3	4100E3	4100E3
TOTAL FIBERS (F/L)	1	1	2300E8	2300E8	2300E8	2300E8	*	1	1	4300E4	4300E4	4300E4	4300E4

Table VI-4

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY COPPER/LEAD/ZINC/GOLD/SILVER/PLATINUM/MOLYBDENUM
SUBDIVISION MINE
MILL PROCESS MINE DRAINAGE

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	15	3	.11633	0.121	0.132	0.132	*	5	0				
ARSENIC (TOTAL)	15	11	.03727	0.018	0.1708	0.196	*	5	2	0.01	0.01	0.01	0.01
BERYLLIUM (TOTAL)	15	4	.00908	.00715	0.022	0.022	*	5	1	0.0064	0.0064	0.0064	0.0064
CADMIUM (TOTAL)	15	9	.02819	0.005	0.124	0.124	*	5	2	.01055	.01055	0.013	0.013
CHROMIUM (TOTAL)	15	7	0.0216	0.017	0.065	0.065	*	5	0				
COPPER (TOTAL)	15	14	.76418	0.045	4.285	7.3	*	5	5	0.056	0.04	0.12	0.12
CYANIDE (TOTAL)	11	2	0.0115	0.0115	0.02	0.02	*	4	1	0.035	0.035	0.035	0.035
LEAD (TOTAL)	15	10	1.0909	0.287	5.496	5.87	*	5	4	.06275	0.066	0.099	0.099
MERCURY (TOTAL)	15	4	0.0027	0.002	0.0063	0.0063	*	5	1	0.049	0.049	0.049	0.049
NICKEL (TOTAL)	15	11	.07084	0.059	0.184	0.2	*	5	2	0.3205	0.3205	0.601	0.601
SELENIUM (TOTAL)	15	1	0.012	0.012	0.012	0.012	*	5	0				
SILVER (TOTAL)	15	5	.01106	0.012	0.02	0.02	*	5	1	0.03	0.03	0.03	0.03
THALLIUM (TOTAL)	15	4	0.1045	0.0715	0.269	0.269	*	5	2	0.309	0.309	0.476	0.476
ZINC (TOTAL)	15	15	5.2516	0.31	23.98	28.15	*	5	5	3.762	0.53	13.99	13.99
COD	12	12	24.289	7.9	118.15	154	*	4	4	27.25	14	77	77
VSS	6	5	16.454	3.2	70	70	*	2	2	2.5	2.5	3	3
TSS	13	13	195.2	20	1094.6	1456	*	5	5	11	10	20	20
TOC	10	10	7.6507	3.75	22.3	23	*	2	2	3.5	3.5	6	6
PH (UNITS)	12	12	7.1125	7.1	8.16	8.25	*	4	4	8.2875	8.225	9	9
PHENOLICS (4AAP)	13	9	.00822	0.008	0.016	0.016	*	4	2	.00755	.00755	0.0101	0.0101
IRON (TOTAL)	7	7	20.05	1.44	133.4	133.4	*	3	3	9.4117	0.65	27.36	27.36
ASBESTOS (CHRYSO) (F/L)	6	6	917E13	3063E6	550E14	550E14	*	2	2	4650E3	4650E3	8200E3	8200E3
TOTAL FIBERS (F/L)	6	5	2424E7	1000E5	1200E8	1200E8	*	2	2	6450E4	6450E4	7200E4	7200E4

Table VI-5

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY COPPER/LEAD/ZINC/GOLD/SILVER/PLATINUM/MOLYBDENUM
SUBDIVISION MILL
MILL PROCESS CYANIDATION

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	2	1	0.1	0.1	0.1	0.1	*						
ARSENIC (TOTAL)	2	2	100	100	200	200	*						
BERYLLIUM (TOTAL)	2	1	0.03	0.03	0.03	0.03	*						
CADMIUM (TOTAL)	2	0					*						
CHROMIUM (TOTAL)	2	2	0.825	0.825	1.6	1.6	*						
COPPER (TOTAL)	2	2	1.44	1.44	2.6	2.6	*						
CYANIDE (TOTAL)	2	2	3.85	3.85	6.8	6.8	*						
LEAD (TOTAL)	2	2	0.195	0.195	0.37	0.37	*						
MERCURY (TOTAL)	2	2	0.2775	0.2775	0.54	0.54	*						
NICKEL (TOTAL)	2	0					*						
SELENIUM (TOTAL)	2	2	0.0775	0.0775	0.15	0.15	*						
SILVER (TOTAL)	2	1	0.1	0.1	0.1	0.1	*						
THALLIUM (TOTAL)	2	0					*						
ZINC (TOTAL)	2	2	2.16	2.16	3.9	3.9	*						
COD	2	2	354	354	700	700	*						
VSS	2	2	649	649	1290	1290	*						
TSS	2	2	30149	30149	60200	60200	*						
TOC	2	2	11.5	11.5	18	18	*						
PH (UNITS)	2	2	8.825	8.825	9	9	*						
PHENOLICS (4AAP)	2	0					*						
ASBESTOS (CHRYSO) (F/L)	2	2	1372E6	1372E6	2700E6	2700E6	*						
TOTAL FIBERS (F/L)	2	2	5527E7	5527E7	1100E8	1100E8	*						

Table VI-6

DATA SUMMARY
ORE MINING DATA

SUBCATEGORY COPPER/LEAD/ZINC/GOLD/SILVER/PLATINUM/MOLYBDENUM
SUBDIVISION MILL
MILL PROCESS FLOTATION (FROTH)

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%	*	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%
ANTIMONY (TOTAL)	78	13	44.7	1725	167	1359	*	59	3	100	133.33	100	170
ARSENIC (TOTAL)	78	78	28	2853.4	800	8100	*	59	43	6	75.244	12.5	285
BERYLLIUM (TOTAL)	78	55	0.35	75.401	75	150	*	59	7	1	5.7143	3	12.1
CADMIUM (TOTAL)	78	41	0.81	637.99	170	1178	*	59	6	5	7.3333	5	11.6
CHROMIUM (TOTAL)	78	63	20	4643.1	1850	11000	*	59	20	10	162.05	30	320
COPPER (TOTAL)	78	78	307.2	98854	63300	292000	*	59	55	20	313.64	70	640
CYANIDE (TOTAL)	74	31	40	282.91	180	590	*	51	12	44	165	120	256
LEAD (TOTAL)	78	69	100.7	20192	2750	27300	*	59	27	20	100.19	42.5	233
MERCURY (TOTAL)	72	48	0.2	51.63	1.1	22.2	*	58	16	0.5	27.3	0.8	68
NICKEL (TOTAL)	78	72	72.8	3708.6	2000	9200	*	59	35	25	90.543	60	185
SELENIUM (TOTAL)	77	50	12	242.38	200	526.67	*	58	23	5	23.225	12.083	34
SILVER (TOTAL)	78	43	11.84	410.99	251.67	805	*	59	8	11	31.375	20	46
THALLIUM (TOTAL)	78	7	1.7	89.557	8.1	197.8	*	59	0				
ZINC (TOTAL)	78	78	98	74137	5600	266400	*	59	48	30	258.12	70	562
TOTAL FIBERS	13	13	7.4E+09	1.8E+12	5.4E+11	3.5E+12	*	14	14	3.7E+06	6.1E+08	1.9E+07	1.9E+09
ASBESTOS (CHRYSOTILE)	15	15	1.6E+09	2.3E+11	4.8E+10	4.2E+11	*	14	14	1.5E+05	2.2E+08	1.7E+06	3.2E+08
COD (MG/L)	22	22	22.2	1126.8	530	2988	*	15	14	4.4	15.655	12	26.9
VSS (MG/L)	9	9	345	10244	3750	14794	*	8	7	.03162	1.7211	1.5	3.15
TSS (MG/L)	22	22	38.6	199538	164000	444199	*	21	20	1	11.483	4.6	17.333
TOC (MG/L)	22	22	3.4667	14.864	9.5	28.6	*	15	15	3	11.667	9.5	20.5
PH (UNITS)	22	22	6.52	8.7848	8.35	11.61	*	21	21	6.21	7.8226	7.825	8.8
PHENOLICS (4AAP)	73	65	11.5	214.54	35.5	402.5	*	52	48	9	74.326	19	216
IRON (TOTAL-MG/L)	9	9	1.9	57.733	28.5	159.53	*	6	6	0.095	.56867	0.15	1.288

Table VI-7

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY COPPER/LEAD/ZINC/GOLD/SILVER/PLATINUM/MOLYBDENUM
SUBDIVISION MINE/MILL
MILL PROCESS HEAP/VAT/DUMP LEACHING

	RAW(MG/L)						*	TREATED (MG/L)					
			DETECTED VALUES ONLY				*			DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	1	0					*	1	0				
ARSENIC (TOTAL)	1	1	0.027	0.027	0.027	0.027	*	1	1	0.002	0.002	0.002	0.002
BERYLLIUM (TOTAL)	1	1	0.3	0.3	0.3	0.3	*	1	1	0.002	0.002	0.002	0.002
CADMIUM (TOTAL)	1	1	0.26	0.26	0.26	0.26	*	1	1	0.003	0.003	0.003	0.003
CHROMIUM (TOTAL)	1	1	1.3	1.3	1.3	1.3	*	1	0				
COPPER (TOTAL)	1	1	88	88	88	88	*	1	1	0.023	0.023	0.023	0.023
LEAD (TOTAL)	1	1	0.01	0.01	0.01	0.01	*	1	1	100E-5	100E-5	100E-5	100E-5
MERCURY (TOTAL)	1	0					*	1	0				
NICKEL (TOTAL)	1	1	14.2	14.2	14.2	14.2	*	1	1	0.028	0.028	0.028	0.028
SELENIUM (TOTAL)	1	0					*	1	0				
SILVER (TOTAL)	1	0					*	1	0				
THALLIUM (TOTAL)	1	1	0.003	0.003	0.003	0.003	*	1	1	0.003	0.003	0.003	0.003
ZINC (TOTAL)	1	1	107	107	107	107	*	1	1	0.013	0.013	0.013	0.013
TSS	1	1	323	323	323	323	*	1	1	50	50	50	50
PH (UNITS)	1	1	3.04	3.04	3.04	3.04	*	1	1	7.87	7.87	7.87	7.87
IRON (TOTAL)	1	1	1860	1860	1860	1860	*	1	0				

Table VI-8

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY COPPER/LEAD/ZINC/GOLD/SILVER/PLATINUM/MOLYBDENUM
SUBDIVISION MINE/MILL
MILL PROCESS GRAVITY SEPARATION

	RAW(MG/L)						*	TREATED (MG/L)					
			DETECTED VALUES ONLY				*			DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ARSENIC (TOTAL)	11	11	1.1736	0.2	4.78	5	*	10	10	0.1729	0.05	1.105	1.2
MERCURY (TOTAL)	11	11	436E-6	100E-6	.00134	0.0014	*	10	10	150E-6	100E-6	470E-6	500E-6
TSS	11	11	18.593	12.4	59.34	64.1	*	10	10	1.4462	0.757	5.45	5.7
PH (UNITS)	6	6	7.2	7.15	7.9	7.9	*	6	6	7.2	7.25	7.9	7.9

Table VI-9

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY ALUMINUM
SUBDIVISION MINE
MILL PROCESS MINE DRAINAGE

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	1	0					*	1	0				
ARSENIC (TOTAL)	1	0					*	1	0				
BERYLLIUM (TOTAL)	1	0					*	1	0				
CADMIUM (TOTAL)	1	0					*	1	0				
CHROMIUM (TOTAL)	1	1	0.03	0.03	0.03	0.03	*	1	1	0.025	0.025	0.025	0.025
COPPER (TOTAL)	1	1	0.06	0.06	0.06	0.06	*	1	1	0.05	0.05	0.05	0.05
CYANIDE (TOTAL)	1	0					*	1	0				
LEAD (TOTAL)	1	0					*	1	0				
MERCURY (TOTAL)	1	1	0.037	0.037	0.037	0.037	*	1	1	0.084	0.084	0.084	0.084
NICKEL (TOTAL)	1	1	0.06	0.06	0.06	0.06	*	1	0				
SELENIUM (TOTAL)	1	0					*	1	0				
SILVER (TOTAL)	1	0					*	1	0				
THALLIUM (TOTAL)	1	0					*	1	0				
ZINC (TOTAL)	1	1	0.57	0.57	0.57	0.57	*	1	0				
COD	1	0					*	1	1	2	2	2	2
VSS	1	1	1.6	1.6	1.6	1.6	*	1	1	5	5	5	5
TSS	1	1	2.8	2.8	2.8	2.8	*	1	1	6	6	6	6
TOC	1	1	2	2	2	2	*	1	1	4	4	4	4
PH (UNITS)	1	1	3.05	3.05	3.05	3.05	*	1	1	8.6	8.6	8.6	8.6
PHENOLICS (4AAP)	2	1	0.005	0.005	0.005	0.005	*	3	2	0.0245	0.0245	0.044	0.044
ASBESTOS (CHRYSO) (F/L)	2	2	5500E3	5500E3	5500E3	5500E3	*	2	2	1016E5	1016E5	2000E5	2000E5
TOTAL FIBERS (F/L)	1	1	3500E4	3500E4	3500E4	3500E4	*	2	2	7500E5	7500E5	1400E6	1400E6

Table VI-10

DATA SUMMARY
ORE MINING DATA

SUBCATEGORY TUNGSTEN
SUBDIVISION MILL

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%	*	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%
ANTIMONY (TOTAL)	2	1	53	53	53	53	*	1	1	3	3	3	3
ARSENIC (TOTAL)	2	1	370	370	370	370	*	1	1	15	15	15	15
BERYLLIUM (TOTAL)	2	2	90	420	90	618	*	1	1	0.7	0.7	0.7	0.7
CADMIUM (TOTAL)	2	2	160	260	160	320	*	1	1	36	36	36	36
CHROMIUM (TOTAL)	2	2	680	940	680	1096	*	1	1	15	15	15	15
COPPER (TOTAL)	2	2	19000	22000	19000	23800	*	1	1	14000	14000	14000	14000
CYANIDE (TOTAL)	1	0					*						
LEAD (TOTAL)	2	2	1300	3050	1300	4100	*	1	1	220	220	220	220
MERCURY (TOTAL)	2	1	2	2	2	2	*	1	0				
NICKEL (TOTAL)	2	2	890	1345	890	1618	*	1	0				
SELENIUM (TOTAL)	2	2	11	30.5	11	42.2	*	1	0				
SILVER (TOTAL)	2	2	210	295	210	346	*	1	1	35	35	35	35
THALLIUM (TOTAL)	2	0					*	1	0				
ZINC (TOTAL)	2	2	10000	17000	10000	21200	*	1	1	1100	1100	1100	1100
TOTAL FIBERS	1	1	1.3E+12	1.3E+12	1.3E+12	1.3E+12	*						
ASBESTOS (CHRYSOTILE)	1	1	3.7E+11	3.7E+11	3.7E+11	3.7E+11	*						
COD (MG/L)	1	1	300	300	300	300	*						
VSS (MG/L)	1	1	4400	4400	4400	4400	*						
TSS (MG/L)	2	2	29000	257000	29000	393800	*	1	1	160	160	160	160
TOC (MG/L)	1	1	220	220	220	220	*						
PH (UNITS)	2	2	9.9	9.9	9.9	9.9	*	1	1	9.2	9.2	9.2	9.2
PHENOLICS (4AAP)	2	1	63	63	63	63	*						
IRON (TOTAL-MG/L)	1	1	660	660	660	660	*	1	1	15	15	15	15

Table VI-11

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY MERCURY
SUBDIVISION MILL
MILL PROCESS FLOTATION (FROTH)

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	1	1	53	53	53	53	*	1	1	0.2	0.2	0.2	0.2
ARSENIC (TOTAL)	1	1	1.1	1.1	1.1	1.1	*	1	1	0.11	0.11	0.11	0.11
BERYLLIUM (TOTAL)	1	1	0.09	0.09	0.09	0.09	*	1	0				
CADMIUM (TOTAL)	1	1	0.56	0.56	0.56	0.56	*	1	1	0.008	0.008	0.008	0.008
CHROMIUM (TOTAL)	1	1	0.46	0.46	0.46	0.46	*	1	1	0.015	0.015	0.015	0.015
COPPER (TOTAL)	1	1	0.85	0.85	0.85	0.85	*	1	1	0.05	0.05	0.05	0.05
CYANIDE (TOTAL)	1	0					*	1	0				
LEAD (TOTAL)	1	1	1	1	1	1	*	1	0				
MERCURY (TOTAL)	1	1	230	230	230	230	*	1	1	0.05	0.05	0.05	0.05
NICKEL (TOTAL)	1	1	1.6	1.6	1.6	1.6	*	1	0				
SELENIUM (TOTAL)	1	0					*	1	0				
SILVER (TOTAL)	1	1	0.01	0.01	0.01	0.01	*	1	0				
THALLIUM (TOTAL)	1	1	0.2	0.2	0.2	0.2	*	1	0				
ZINC (TOTAL)	1	1	2.4	2.4	2.4	2.4	*	1	1	0.04	0.04	0.04	0.04
COD	1	1	60	60	60	60	*	1	1	22	22	22	22
VSS	1	1	4300	4300	4300	4300	*	1	0				
TSS	1	1	139000	139000	139000	139000	*	1	1	16	16	16	16
TOC	1	1	21	21	21	21	*	1	1	13	13	13	13
PH (UNITS)	1	1	8	8	8	8	*	1	1	8.3	8.3	8.3	8.3
PHENOLICS (4AAP)	1	1	0.92	0.92	0.92	0.92	*	1	1	0.22	0.22	0.22	0.22
ASBESTOS (CHRYSO) (F/L)	1	1	1500E8	1500E8	1500E8	1500E8	*	1	1	5700E4	5700E4	5700E4	5700E4
TOTAL FIBERS (F/L)	1	1	1300E9	1300E9	1300E9	1300E9	*	1	1	7700E5	7700E5	7700E5	7700E5

Table VI-12

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY URANIUM
SUBDIVISION MINE
MILL PROCESS MINE DRAINAGE

	RAW(MG/L)						*	TREATED (MG/L)					
			DETECTED VALUES ONLY				*			DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	3	1	0.05	0.05	0.05	0.05	*	3	0				
ARSENIC (TOTAL)	17	16	0.0195	0.007	0.0832	0.17	*	13	11	.00798	0.006	0.0228	0.024
BERYLLIUM (TOTAL)	3	0					*	3	0				
CADMIUM (TOTAL)	16	13	.00381	0.003	0.0092	0.01	*	13	10	0.0038	0.003	0.0069	0.007
CHROMIUM (TOTAL)	4	3	.04333	0.045	0.05	0.05	*	3	2	0.0425	0.0425	0.06	0.06
COPPER (TOTAL)	14	14	.01673	0.0075	0.075	0.11	*	11	8	.00575	0.006	0.011	0.011
CYANIDE (TOTAL)	3	0					*	3	0				
LEAD (TOTAL)	4	3	0.09	0.05	0.18	0.18	*	3	1	0.05	0.05	0.05	0.05
MERCURY (TOTAL)	3	1	0.0038	0.0038	0.0038	0.0038	*	3	1	0.0091	0.0091	0.0091	0.0091
NICKEL (TOTAL)	4	1	0.06	0.06	0.06	0.06	*	3	0				
SELENIUM (TOTAL)	5	3	.02333	0.028	0.037	0.037	*	5	3	.03633	0.048	0.051	0.051
SILVER (TOTAL)	3	0					*	3	0				
THALLIUM (TOTAL)	3	0					*	3	0				
ZINC (TOTAL)	17	17	.04306	0.02	0.158	0.19	*	13	12	.01983	0.014	0.0666	0.078
COD	15	15	22.504	7	104.2	140.5	*	12	12	10.169	8.95	33.5	38
VSS	2	2	23.5	23.5	28	28	*	2	2	1.5	1.5	2	2
TSS	18	18	144.58	21	415.94	1639.5	*	13	13	33.185	27	75.8	83
TOC	2	2	8.5	8.5	9	9	*	2	1	10	10	10	10
PH (UNITS)	13	13	7.6519	8.05	8.655	8.825	*	9	9	7.8833	7.9	8.5	8.5
PHENOLICS (4AAP)	3	1	0.01	0.01	0.01	0.01	*	3	1	0.01	0.01	0.01	0.01
IRON (TOTAL)	1	1	0.319	0.319	0.319	0.319	*	1	1	0.054	0.054	0.054	0.054
ASBESTOS (CHRYSO) (F/L)	3	3	1050E5	1100E5	1900E5	1900E5	*	2	2	4000E4	4000E4	5300E4	5300E4
TOTAL FIBERS (F/L)	2	2	1950E6	1930E6	2300E6	2300E6	*	2	2	5000E5	5000E5	5700E5	5700E5

Table VI-13

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY URANIUM
SUBDIVISION MILL
MILL PROCESS ARID LOCATIONS

	RAW(MG/L)						*	TREATED (MG/L)					
				DETECTED VALUES ONLY			*				DETECTED VALUES ONLY		
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	4	2	0.516	0.516	1.03	1.03	*	5	3	0.299	100E-5	0.895	0.895
ARSENIC (TOTAL)	10	9	4.2602	0.243	10.6	10.6	*	12	11	.11518	0.029	0.65	0.75
BERYLLIUM (TOTAL)	6	2	0.274	0.274	0.295	0.295	*	7	3	0.0072	0.01	0.011	0.011
CADMIUM (TOTAL)	12	11	.14791	0.1	0.4068	0.423	*	13	11	.03545	0.029	0.0746	0.077
CHROMIUM (TOTAL)	8	8	1.738	1.575	3.7	3.7	*	10	5	0.0406	0.028	0.1	0.1
COPPER (TOTAL)	12	10	0.9966	0.485	3.4	3.4	*	14	11	0.192	0.1	0.84	0.9
CYANIDE (TOTAL)	2	1	0.046	0.046	0.046	0.046	*	3	0				
LEAD (TOTAL)	8	5	1.9076	1.3	4.18	4.18	*	10	5	0.3888	0.2	0.959	0.959
MERCURY (TOTAL)	4	1	0.036	0.036	0.036	0.036	*	5	1	0.14	0.14	0.14	0.14
NICKEL (TOTAL)	8	8	2.3422	2.835	3.68	3.68	*	10	8	.82637	0.955	1.28	1.28
SELENIUM (TOTAL)	6	6	0.1705	0.1525	0.49	0.49	*	7	6	.06383	0.0215	0.213	0.213
SILVER (TOTAL)	6	3	0.069	0.056	0.1	0.1	*	7	4	0.016	0.0195	0.023	0.023
THALLIUM (TOTAL)	4	2	1.205	1.205	1.24	1.24	*	5	2	0.79	0.79	0.84	0.84
ZINC (TOTAL)	12	12	26.176	22.365	59.13	60.9	*	14	13	4.729	2.52	11.06	11.1
COD	5	5	95.206	26	386	386	*	7	6	59.505	10.5	279	279
VSS	1	1	20	20	20	20	*	2	2	6	6	10	10
TSS	5	5	19134	64	95450	95450	*	9	9	55.611	26	157	157
TOC	1	1	24	24	24	24	*	2	2	21.5	21.5	27	27
PH (UNITS)	6	6	6.43	7.45	8.3	8.3	*	9	9	6.65	7.7	8.45	8.45
PHENOLICS (4AAP)	2	2	0.0085	0.0085	0.01	0.01	*	3	2	0.01	0.01	0.01	0.01
IRON (TOTAL)	7	5	1462.1	1660	2040	2040	*	7	5	1.4164	0.4	3.87	3.87
ASBESTOS (CHRYSO) (F/L)	1	1	2300E4	2300E4	2300E4	2300E4	*	2	2	1750E5	1750E5	2000E5	2000E5
TOTAL FIBERS (F/L)	1	1	2900E5	2900E5	2900E5	2900E5	*	2	2	1750E6	1750E6	2300E6	2300E6

Table VI-14

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY TITANIUM
SUBDIVISION MINE
MILL PROCESS MINE DRAINAGE

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	DETECTED VALUES ONLY MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	DETECTED VALUES ONLY MEDIAN	90%	MAX
ANTIMONY (TOTAL)							*	1	0				
ARSENIC (TOTAL)							*	1	0				
BERYLLIUM (TOTAL)							*	1	0				
CADMIUM (TOTAL)							*	1	0				
CHROMIUM (TOTAL)							*	1	0				
COPPER (TOTAL)							*	1	1	0.02	0.02	0.02	0.02
CYANIDE (TOTAL)							*	1	0				
LEAD (TOTAL)							*	1	1	0.02	0.02	0.02	0.02
MERCURY (TOTAL)							*	1	0				
NICKEL (TOTAL)							*	1	0				
SELENIUM (TOTAL)							*	1	0				
SILVER (TOTAL)							*	1	0				
THALLIUM (TOTAL)							*	1	0				
ZINC (TOTAL)							*	1	1	0.02	0.02	0.02	0.02
COD							*	1	1	2	2	2	2
VSS							*	1	0				
TSS							*	1	0				
TOC							*	1	1	8	8	8	8
PH (UNITS)							*	1	1	7.95	7.95	7.95	7.95
PHENOLICS (4AAP)							*	1	1	0.03	0.03	0.03	0.03
ASBESTOS (CHRYSO) (F/L)							*	1	1	140000	140000	140000	140000
TOTAL FIBERS (F/L)							*	1	1	1900E3	1900E3	1900E3	1900E3

Table VI-15

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY TITANIUM
SUBDIVISION MILLS WITH DREDGE MINING
MILL PROCESS PHYSICAL AND/OR CHEMICAL

	RAW(MG/L)						*	TREATED (MG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	9	1	0.002	0.002	0.002	0.002	*	9	0				
ARSENIC (TOTAL)	9	3	.00867	0.009	0.01	0.01	*	9	0				
BERYLLIUM (TOTAL)	9	0					*	9	0				
CADMIUM (TOTAL)	9	0					*	9	1	0.002	0.002	0.002	0.002
CHROMIUM (TOTAL)	9	7	.04743	0.03	0.08	0.08	*	9	0				
COPPER (TOTAL)	9	9	.02733	0.016	0.063	0.063	*	9	5	0.0058	0.006	0.008	0.008
CYANIDE (TOTAL)	6	0					*	9	0				
LEAD (TOTAL)	9	4	0.0375	0.042	0.058	0.058	*	9	1	0.005	0.005	0.005	0.005
MERCURY (TOTAL)	9	2	0.006	0.006	0.011	0.011	*	9	1	100E-5	100E-5	100E-5	100E-5
NICKEL (TOTAL)	9	3	0.023	0.023	0.033	0.033	*	9	0				
SELENIUM (TOTAL)	9	3	0.031	0.029	0.036	0.036	*	9	0				
SILVER (TOTAL)	9	5	0.007	0.009	0.011	0.011	*	9	2	0.003	0.003	0.003	0.003
THALLIUM (TOTAL)	9	0					*	9	0				
ZINC (TOTAL)	9	9	.03122	0.021	0.071	0.071	*	9	8	.02675	0.008	0.071	0.071
COD	6	6	1076.7	1060.5	1900	1900	*	9	9	14	14	17	17
TSS	9	9	341.44	160	1100	1100	*	9	8	3.5625	2.9	9	9
TOC	6	6	485	560	750	750	*	9	6	5.1875	5.25	6	6
PH (UNITS)	9	9	5.9	5.7	6.6	6.8	*	9	9	5.9444	6.8	7.8	7.8
PHENOLICS (4AAP)	6	5	0.0066	0.007	0.007	0.007	*	8	1	0.01	0.01	0.01	0.01
IRON (TOTAL)	9	9	3.1924	1.928	6.287	6.287	*	9	9	.20533	0.171	0.5	0.5
ASBESTOS (CHRYSO) (F/L)							*	1	1	3300E3	3300E3	3300E3	3300E3
TOTAL FIBERS (F/L)							*	1	1	2700E3	2700E3	2700E3	2700E3

Table VI-16

DATA SUMMARY
ORE MINING DATA

SUBCATEGORY VANADIUM
SUBDIVISION MINE
MILL PROCESS NO MILL PROCESS

	RAW(UG/L)							TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%	*	NUMBER OF SAMPLES	NUMBER DETECTED	10%	MEAN	MED	90%
ANTIMONY (TOTAL)	1	1	18	18	18	18	*	1	1	2	2	2	2
ARSENIC (TOTAL)	1	1	130	130	130	130	*	1	1	5	5	5	5
BERYLLIUM (TOTAL)	1	1	16.333	16.333	16.333	16.333	*	1	1	1	1	1	1
CADMIUM (TOTAL)	1	1	16.833	16.833	16.833	16.833	*	1	1	8.2	8.2	8.2	8.2
CHROMIUM (TOTAL)	1	1	120	120	120	120	*	1	1	29.333	29.333	29.333	29.333
COPPER (TOTAL)	1	1	41.833	41.833	41.833	41.833	*	1	1	20.5	20.5	20.5	20.5
CYANIDE (TOTAL)	1	0					*	1	0				
LEAD (TOTAL)	1	1	317.5	317.5	317.5	317.5	*	1	1	171.33	171.33	171.33	171.33
MERCURY (TOTAL)	1	1	1	1	1	1	*	1	0				
NICKEL (TOTAL)	1	1	446.67	446.67	446.67	446.67	*	1	1	59.333	59.333	59.333	59.333
SELENIUM (TOTAL)	1	1	6.3333	6.3333	6.3333	6.3333	*	1	1	12	12	12	12
SILVER (TOTAL)	1	1	3	3	3	3	*	1	0				
THALLIUM (TOTAL)	1	1	2	2	2	2	*	1	1	1	1	1	1
ZINC (TOTAL)	1	1	1476.7	1476.7	1476.7	1476.7	*	1	1	159.17	159.17	159.17	159.17
PHENOLICS (4AAP)	1	0					*	1	0				
IRON (TOTAL-MG/L)	1	1	69.133	69.133	69.133	69.133	*	1	1	.86467	.86467	.86467	.86467

Table VI-17

DATA SUMMARY
ORE MINING DATA
SUBCATEGORY VANADIUM
SUBDIVISION MILL
MILL PROCESS FLOTATION (FROTH)

	RAW(MG/L)						*	TREATED (MG/L)					
			DETECTED VALUES ONLY				*			DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	3	3	.03733	.04167	.05233	.05233	*	3	3	.02067	0.014	0.046	0.046
ARSENIC (TOTAL)	3	3	.29889	.37333	.39333	.39333	*	3	3	0.108	0.014	0.305	0.305
BERYLLIUM (TOTAL)	3	3	0.041	0.038	.06867	.06867	*	3	3	.05317	0.036	0.1225	0.1225
CADMIUM (TOTAL)	3	3	.18456	0.0245	.51233	.51233	*	3	3	.02384	0.025	.03833	.03833
CHROMIUM (TOTAL)	3	3	1.6493	.47117	4.3567	4.3567	*	3	3	.14189	.06133	0.335	0.335
COPPER (TOTAL)	3	3	4.2113	.06533	12.527	12.527	*	3	3	.03869	.04225	.04733	.04733
CYANIDE (TOTAL)	3	1	0.29	0.29	0.29	0.29	*	3	0				
LEAD (TOTAL)	3	3	5.7773	0.3175	16.717	16.717	*	3	3	.55928	0.3265	1.18	1.18
MERCURY (TOTAL)	3	2	0.1425	0.1425	0.284	0.284	*	3	0				
NICKEL (TOTAL)	3	3	.78883	.44667	1.8207	1.8207	*	3	3	.13636	.09475	0.255	0.255
SELENIUM (TOTAL)	3	3	.86767	0.14	2.4567	2.4567	*	3	3	.08367	0.079	0.16	0.16
SILVER (TOTAL)	3	3	.03089	.02667	0.063	0.063	*	3	2	0.009	0.009	0.016	0.016
THALLIUM (TOTAL)	3	3	.29211	0.002	.87333	.87333	*	3	3	.11667	0.0015	0.3475	0.3475
ZINC (TOTAL)	3	3	34.165	1.4767	100.82	100.82	*	3	3	.11664	0.1115	.15917	.15917
PHENOLICS (4AAP)	3	0					*	3	0				
IRON (TOTAL)	3	3	135.24	69.133	334.9	334.9	*	3	3	.55572	0.5505	.86467	.86467

TABLE VI-18 SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS*

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
		MODIFIERS		
Lime	CaO or $Ca(OH)_2$	Alkaline pH regulator and depressant for galena, metallic gold, iron sulfides, cobalt, and nickel sulfide. Has flocculating effect on ore slimes.	15	0.054 - 14.2
Caustic soda	Sodium Hydroxide, NaOH	Alkaline pH regulator	3	0.00015 - 0.025
Soda Ash	Sodium Carbonate, Na_2CO_3	Alkaline pH regulator w/slime dispersing action.	3	0.54 - 12.12
Sulfuric Acid	H_2SO_4	Acidic pH regulator.	2	0.018 - 4.3
		ACTIVATORS		
Copper Sulfate	$CuSO_4$ or $CuSO_4 \cdot 5H_2O$	Universal activator for sphalerite. Also used for the reactivation of minerals depressed by cyanide.	13	0.06 - 2.32
Copper Ammonium Chloride	$CuNH_2Cl$	Activator for sphalerite. Purchased as a waste by-product from the manufacture of electric circuit boards.	1	0.13 †
Sodium Sulfhydrate	Sodium Hydrosulfide $NaSH \cdot 2H_2O$	Activator for copper sulfide minerals.	1	0.0094

*Copper, lead, zinc, silver, and molybdenum concentrators which discharge process wastewater (data available).

† Expressed as soluble copper metal.

**Reagent usage data supplied by 22 milling operations.

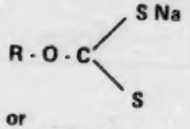
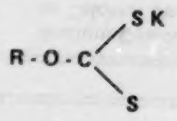
TABLE VI-18. SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS*
(Continued)

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
		DEPRESSANTS		
Cyanide	Sodium Cyanide, NaCN or Calcium Cyanide, $Ca(CN)_2$	Strong depressants for the iron sulfides, arsenopyrite, and sphalerite. Mild depressant for chalcopyrite, enargite, bornite and most other sulfide minerals w/ exception of galena.	13	0.003 - 0.065
Sodium Sulfite	$NaSO_3$	Depressant for pyrite and sphalerite while floating lead and/or copper.	2	0.2 - 7.46
Zinc Sulfate	$ZnSO_4 \cdot H_2O$ or $ZnSO_4 \cdot 7H_2O$	Depressant for sphalerite while floating lead and/or copper minerals. Often used in conjunction w/ cyanide.	7	0.1 - 1.35
Sodium Dichromate	$Na_2Cr_2O_7$	Depressant for galena in copper-lead separations. Excess depresses copper sulfides and iron sulfides.	1	0.022
Dextrin Starch Guar Hodag-31	Corn starch	Depressant for galena and molybdenite while floating copper sulfides	4	0.0005 - 0.071
Sulfur Dioxide	SO_2	Depressant for galena and activator for copper sulfides. Often used in conjunction w/ starch.	2	0.156 - 0.406
Noke's Reagent	Phosphorus Pentasulfide P_2S_5	Depressant for copper and lead while floating molybdenite.	4	0.0001 - 0.47
Hydrogen Peroxide	H_2O_2	Depressant for copper sulfides in copper-molybdenite separations.	1	0.016

*Copper, lead, zinc, silver, and molybdenum concentrators which discharge process wastewater (data available).

**Reagent usage data suplied by 22 milling operations.

TABLE VI-18. SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS* (Continued)

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
		DEPRESSANTS		
Sodium Silicate	$Na_2O: nSiO_2$	Depressant for quartz and other siliceous gangue minerals. Also acts as slime dispersant.	5	0.031 - 2.08
AERO Depressant 610, 633	Composition unknown – Contains ~ 1.5% phenolics	Depressant for graphitic and talcose gangue. Also acts as gangue dispersants useful in sand-slime separation.	3	0.001 - 0.16
Jaguar Mud	Colloidal material	Depressant for gangue materials	1	0.016
		COLLECTORS/PROMOTERS		
Xanthates: AERO 301, 325, 343, 355 Dow Z-3, Z-4, Z-6, Z-11, Z-14.	Sodium or potassium salts of xanthic acid.  or  where R is an alkyl group of 2-6 carbon atoms.	Strong promoters for all sulfide minerals. Essentially non-selective in the absence of modifiers.	17	0.0003 - 0.40

*Copper, lead, zinc, silver, and molybdenum concentrators which discharge process wastewater (data available).

**Reagent usage data supplied by 22 milling operations.

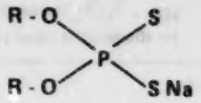
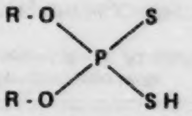
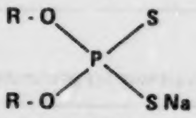
TABLE VI-18. SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS*
(Continued)

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
Dow Z-200	Isopropyl Ethyl-Thionocarbamate	Promoter for copper sulfides and activated sphalerite w/ selectivity over iron sulfides.	3	0.004 - 0.10
Fuel Oil Vapor Oil Tar Oil	Saturated Hydrocarbons	Promoters, usually used for readily floatable minerals, such as molybdenite.	4	0.0013 - 0.78
Minerec	Composition Unknown	Promoter.	1	0.01
DRESSENATE TX-65W SOAP	Composition Unknown	Promoter.	1	0.41
		FROTHERS		
M.I.B.C. (Methyl Isobutyl Carbinol)	Synonomous with Methyl Amyl Alcohol $(CH_3)_2\,CHCH_2CHOHCH_3$	Alcohol type frothers are used for the flotation of sulfide minerals where a selective, fine textured froth is desired.	10	0.008 - 0.17
Methanol	CH_3OH	Frother	1	0.00005
Pine Oil	Composed primarily of terpene hydrocarbons, terpene ketones, and terpene alcohols.	Frother, widely used in sulfide flotation. It exhibits some collecting properties, especially for such readily floatable minerals as talc, graphite and molybdenite. Pine oil produces a tough, persistent froth and has a tendency to float gangue.	5	0.015 - 0.175
Cresylic Acid	Higher homologs of phenol, $C_6H_5 \cdot OH$, particularly cresols, $CH_3 \cdot C_6H_4 \cdot OH$, and xylenols, $C_2H_5 \cdot C_6H_4 \cdot OH$, or $(CH_3)_2 \cdot C_6H_4 \cdot OH$	A powerful frother exhibiting some collecting properties. Produces froth of variable texture and persistence, and tends to be non-selective.	3	0.003 - 0.034

*Copper, lead, zinc, silver, and molybedenum concentrators which discharge process wastewater (data available).

**Reagent usage data supplied by 22 milling operations.

TABLE VI-18 SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS* (Continued)

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
Aliphatic Dithiophosphates: Sodium AEROFLOAT AEROFLOAT 211, 249, 3477	 where R is an alkyl group of 2-6 carbon atoms.	Promoters of variable selectivity, and strength for the flotation of sulfide materials. Sometimes used in conjunction with xanthates for improved precious metal recoveries.	5	0.015 - 0.043
AEROFLOAT 31 and 242	Aryl Dithiophosphoric Acids  where R is an aryl group (benzene-based).	Promoters for copper, lead, zinc and silver sulfide minerals. Has frothing properties.	4	0.012 - 0.05
"Reco"	Sodium Dicresyl Dithiophosphate  where R is the cresyl group: $CH_3 - C_6H_3 - OH$	Promoter, selective to copper sulfide minerals. Very similar to AEROFLOAT 31 and 242.	1	0.016
ARMAC "C"	Acetate Salt of Aliphatic Amines	Promoter, very selective cationic collector.	1	0.0005

*Copper, lead, zinc, silver, and molybdenum concentrators which discharge process wastewater (data available).

**Reagent usage data supplied by 22 milling operations.

TABLE VI-18 SUMMARY OF REAGENT USE (BY FUNCTION) IN ORE FLOTATION MILLS* (Continued)

REAGENT	DESCRIPTION	FUNCTION	NUMBER OF MILLS WHERE USED**	USUAL DOSAGE kg/metric ton ore feed
Polyglycols: DOWFROTH 200, 250 AEROFROTH 65	Polyglycol Methyl Ethers (i.e. Polypropylene glycol methyl ether) $CH_3 \cdot (OC_3H_6)_x \cdot OH$	Frothers, for metallic flotation, w/ froth persistancy and selectivity against nonmetals.	8	0.002 - 0.17
Diphenyl Guanidine	$HN \cdot C\ (NHC_6H_5)_2$	Frother	1	0.00005
UCON-R-23	Composition unknown	Frother	1	0.035
UCON-R-133	Composition unknown	Frother	1	0.015
SYNTEX	Composition unknown	Frother	1	0.017
		FLOCCULANTS		
AEROFLOC AERODRI 100 VALCO 1801 NALCOLYTE 670 SEPARAN NP-10 SUPERFLOC 3302 Flocculants (unspecified)	Anionic, Cationic, or Nonionic Organic Polymers.	Used as dewatering aids or filtration aids for thickening or filtering ore pulps, concentrates, and tailings.	9	0.00015 - 0.051

*Copper, lead, zinc, silver, and molybdenum concentrators which discharge process wastewater (data available).

**Reagent usage data supplied by 22 milling operations.

SECTION VII

SELECTION OF POLLUTANT PARAMETERS

The Agency has studied ore mining and dressing wastewaters to determine the presence or absence of toxic, conventional and non-conventional pollutants. The toxic pollutants are of primary concern to the development of BAT effluent limitations guidelines. One hundred and twenty-nine pollutants (known as the 129 priority pollutants) were studied pursuant to the requirements of the Clean Water Act of 1977 (CWA). The 129 priority pollutants are included in the 65 classes of toxic pollutants referred to in Table 1, Section 307(a)(1) of the CWA.

EPA conducted sampling and analysis at facilities where BPT technologies are in place; therefore, any of the priority pollutants present in treated effluent discharges are subject to regulation by BAT effluent limitations guidelines. The Settlement Agreement in Natural Resources Defense Council, Incorporated, v. Train, 8 ERC 2120 (D.D.C. 1976), modified 12 ERC 1833 (D.D.C. 1979) provides a number of provisions for the exclusion of particular pollutants, categories and subcategories. The criteria for exclusion of pollutants are summarized below:

1. Equal or more stringent protection is already provided by an effluent limitation and guideline promulgated pursuant to Section(s) 301, 304, 306, 307(a), or 307(c) of the CWA.

2. The pollutant is present in the effluent discharge solely as a result of its presence in the intake water taken from the same body of water into which it is discharged.

3. The pollutant is not detectable in the effluent within the category by approved analytical methods or methods representing the state-of-the-art capabilities.

4. The pollutant is detected in only a small number of sources within the category and is uniquely related to only those sources.

5. The pollutant is present in only trace amounts and is neither causing nor likely to cause toxic effects.

6. The pollutant is present in amounts too small to be effectively reduced by technologies known to the Administrator.

7. The pollutant is effectively controlled by the technologies upon which are based other effluent limitations and guidelines.

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DATA BASE

Table VII-1 presents a summary of the data gathered for this study. The sources of data are screen sampling, verification sampling, verification monitoring, EPA Regional sampling, engineering cost site visits, gold placer mining study, titanium sand dredges study, uranium study, and the solid waste study. These data are presented in complete form in Supplement A. The summary table and extensive information about the sampled industries, based on the criteria listed above, are used to determine which pollutant parameters are excluded from regulation.

SELECTED TOXIC PARAMETERS

Several conventional and non-conventional pollutants were found at all the facilities sampled; the 129 priority pollutants occurred on a less frequent basis. The 13 metals listed as priority pollutants, cyanide and asbestos were found at many of the facilities. Six of the 13 metals were detected at levels too low to be effectively reduced by the technologies known to the Administrator. Eighty-seven (87) priority organic pollutants were not found in the treated effluents during sampling. Of the remaining 27 organic pollutants, 17 were found in the effluent of only one or two sources and always at or below 10 ug/l, nine pollutants were detected at levels too low to be effectively reduced by technologies known to the Administrator, and one was uniquely related to the source at which it was found.

The priority pollutants which were identified for controll by BAT include arsenic, asbestos, cadmium, copper, lead, mercury, nickel, zinc, and cyanide. The conventional parameters to be regulated are pH and TSS. The non-conventional parameters are COD, total and dissolved iron, radium 226 (dissolved and total), ammonia, aluminum, and uranium. The priority pollutants, conventionals, and non-conventionals for control in BAT are displayed in Table VII-2. All 114 of the toxic organic pollutants were excluded from regulation. The toxic metals were excluded on a case-by-case basis within certain subcategories and subdivisions. The reasons for exclusion are displayed in Table VII-3.

EXCLUSION OF TOXIC POLLUTANTS THROUGHOUT THE ENTIRE CATEGORY

Pollutants Not Detected by Approved Methods

The toxic organic compounds are primarily synthetic and are not naturally associated with metal ore. As shown in Table VII-1, 27 of the 114 toxic organics were detected during sampling, while 87 toxic organics were not detected in treated wastewater during sampling. Therefore, the 87 toxic organics not detected are excluded by Criterion 3 (the pollutant is not detectable by approved analytical methods).

Of the 27 toxic organics detected, 17 were detected at at least one facility and always at or below 10 ug/l, which is the limit of detection set by the Agency for the toxic organics in these sampling and analysis programs.

1. Chlorobenzene
2. 1,1,1-Trichloroethane
3. Dichlorobromomethane
4. Chloroform
5. Fluorene
6. Ethylbenzene
7. Trichlorofluoromethane
8. Diethyl Phthalate
9. Tetrachloroethylene
10. Toluene
11. -BHC (Alpha)
12. -BHC (Beta)
13. -BHC (Delta)
14. Aldrin
15. Dieldrin
16. Endrin
17. Heptachlor

Thus, it follows that these 17 compounds are subject to exclusion under Criterion 3.

Pollutants Detected But Present in Amounts Too Low to be Effectively Reduced by Known Technologies

Toxic Organic Pollutants

There were 10 organic pollutants detected during the nine sampling programs discussed in Section V at levels above 10 ug/l. In general, the concentrations of nine of these pollutants are so low that they cannot be substantially reduced. In some cases this is because no technologies are known to further reduce them beyond BPT; in other cases, the pollutant reduction cannot be accurately quantified because the analytical error at these low levels can be larger than the value itself. The following nine pollutants are thus excluded from regulation because they were present in amounts too low to be effectively reduced by technologies known to the Administrator (Criterion 6):

(1) Benzene
(2) 1,2-Trans-Dichloroethylene
(3) Phenol
(4) Bis(2-Ethylhexyl) Phthalate
(5) Butyl Benzyl Phthalate
(6) Di-n-Butyl Phthalate
(7) Di-n-Octyl Phthalate
(8) Dimethyl Phthalate
(9) Methylene Chloride

In addition, contamination during sample collection and analysis has been documented for particular organic pollutants including these nine, as discussed below.

Six of the 10 toxic organics detected are members of the phthalate and phenolic classes. During sample collection, automatic composite samplers were equipped with polyvinyl chloride (Tygon) tubing or original manufacturer supplied tubing.

Phthalates are widely used as plasticizers to ensure that the Tygon tubing remains soft and flexible (References 1 and 2). These compounds, added during manufacturing, have a tendency to migrate to the surface of the tubing and leach into water passing through the sampler tubing. In addition, laboratory experiments were performed to determine if phthalates and other priority pollutants could be leached from tubing used on composite samplers. The types of tubing used in these experiments were:

1. Clear tubing originally supplied with the sampler at the time of purchase

2. Tygon S-50-HL, Class VI (replacement tubing)

Results of analysis of the extracts representing the original and replacement Tygon tubing are summarized in Table VII-4. The data indicate that both types contain bis(2-ethylhexyl) phthalate and the original tubing leaches high concentrations of phenol. Although bis(2-ethylhexyl) phthalate was the only phthalate detected in the tubing in these experiments, a similar experiment conducted as part of a study pursuant to the development of BAT Effluent Limitations Guidelines for the Textiles Point Source Category found dimethyl phthalate, diethyl phthalate, di-n-butyl phthalate, and bis(2-ethylhexyl) phthalate in tubing "blanks" (Reference 3).

Three of the volatile organic compounds (benzene, 1,2-transdichloro-ethylene, and methylene chloride) were detected as a result of the analysis of grab samples. The volatile nature of these compounds suggests contamination as a possible source, especially considering the relatively low concentrations detected in the samples. More importantly, all of the compounds may be found in the laboratory as solvents, extraction agents or aerosol propellants. Thus, the presence and/or use of the compounds in the laboratory may be responsible for sample contamination. This type of contamination has been addressed in other studies (Reference 4). In a review of a set of volatile organic blank analytical data, inadvertent contamination was shown to have occurred; the prominent compounds were benzene, toluene, and methylene chloride.

The contamination by the volatiles as discussed above may be due to the changing physical environment during the collection of samples. The volatile sample is collected in a 45- to 125-ml vial. During collection in the field, the sample vial is filled completely with the wastewater, sealed (so that no air is present in the vial at that moment) and chilled to 4 C until the time of analysis. The volume of the water sample will decrease as it cools from ambient conditions to 4 C, inducing an internal pressure in the vial less than atmospheric. In addition, teflon chips were used as lid liners to prevent contamination of the sample by any compounds present in the lid. Experience in the field has shown that it is difficult to ensure a tight seal at the time of collection because the teflon is not pliable. The combination of the poor seal and the formation of the vacuum may encourage contamination from the ambient laboratory atmosphere where, as previously mentioned, volatile organic compounds are prevalent. Methylene chloride, in particular, is used in the analytical procedure as a solvent (References 4, 5); this may explain the detection of and high concentrations of this volatile in 10 to 25 of the treated water samples (Table VII-1).

The presence of the three volatile organic compounds may be attributed to sampling and analytical contamination, and as such, they cannot be conclusively identified with the wastewater.

Toxic Metal Pollutants

Six toxic metal pollutants were detected during the nine sampling programs. Like the toxic organic pollutants, the concentrations of these pollutants were so low that they cannot be substantially reduced by known technologies. Each of the six metals is discussed in more detail below.

Antimony. Antimony removal is discussed in Section VIII and in a report by Hittman Associates (Reference 6). The conclusion of these discussions is that antimony is very difficult to remove in wastewater treatment. Using seven state-of-the-art technologies, Hittman Associates could not attain lower than 500 ug/l in the effluent. Table VII-1 indicates that the maximum concentration observed in effluent from this category was 200 ug/l. Therefore, Criterion 6 (the pollutant is present in amounts too small to be effectively reduced by known technologies) is applicable and antimony is excluded from regulation in this category.

Thallium. Thallium removal is discussed in a report prepared examining the analysis protocol for this toxic metal (Reference 7). The conclusion that may be drawn from this discussion is that the procedure used in the analysis of thallium is subject to interferences which prevent its conclusive identification in wastewater samples. Thallium is, therefore, excluded from BAT regulation since it cannot be conclusively identified in wastewater samples by approved analytical procedures (Criterion 3).

Selenium. There are little data in the literature on selenium removal from industrial wastewater, treatment methods for selenium wastes, or costs associated with removal of selenium from industrial wastewater (Reference 8). Selenium is present in trace amounts in metallic sulfide ores. Generally the selenium is released in the smelting and refining process and is not liberated during mining and milling. Although 37 samples of 73 contained detectable selenium, the mean value reported was 0.059 mg/l. Ninety percent of the samples in which selenium was detected contained 0.112 mg/l or less. Most of the samples contained very low levels of selenium as indicated by a median value of only 0.015 mg/l. No specific treatment data or application of specific treatment could be found in the ore mining and milling industry. Pilot-scale treatment studies have reported removals ranging from 10 to 84 percent using conventional technologies, but only cation and anion exchange used in combination achieved high removal efficiencies (Reference 8). The removals obtained by utilizing conventional technology are not consistent enough to base regulations upon them, and cation and anion exchange, while possibly providing additional treatment, are judged too costly for this industry. Consequently, selenium is excluded from regulation since it is found at levels too low to be effectively reduced by known technologies (Criterion 6).

Silver. Most data available on treatment of waste streams containing silver represent attempts at recovery of this valuable metal from the photographic and electroplating industries. In these industries there has been an ample economic incentive to develop recovery/removal technology because: (1) the silver is valuable and may be reused; and (2) the concentration levels present favor the economic recovery of silver from the waste stream. Four basic methods for silver removal from wastewater are discussed in Reference 8: (1) precipitation, (2) ion exchange, (3) reductive exchange, and (4) electrolytic recovery. Levels to 0.1 mg/l have been reported by various investigators but most of these in bench- or pilot-scale systems. In addition, waste streams in this industry are high in solids, effluent flow rates are very high, and treated effluent levels are already low (mean of treated effluent samples 0.015 mg/l; maximum 0.04). It has been concluded that the concentration levels present are too low to be effectively reduced by known technologies (Criterion 6).

Beryllium. There are little data available in the literature for beryllium removal in wastewater from the ore mining and dressing industry. Only one domestic facility mines beryllium ore and uses water in a beneficiation process (Mine/Mill 9902). This mill uses a proprietary leach process with raw wastewater having beryllium at a concentration of 36 mg/l at pH 2.6 (Reference 9). This facility has no discharge. However, when TSS in the impoundment is reduced from 116,000 to 44,000 mg/l (after a short settling period), beryllium is reduced to 25 mg/l. Since beryllium is relatively insoluble, it is believed that reduction to an effluent level of TSS of 20 mg/l after lime precipitation and settling would result in a substantial reduction of beryllium levels.

In a related industry, primary beryllium refining, some data are available which indicate effluent beryllium levels of 0.09 mg/l are possible by lime precipitation and multiple pond settling (Reference 10). Of 73 effluent samples analyzed during BAT screening for beryllium, only 10 samples had detectable beryllium concentrations with a mean of 0.005 mg/l and maximum concentration of 0.011 mg/l. These are the levels which are achieved by known technologies and since the pollutant is present only in trace amounts, it has been concluded that the present concentration level would not be effectively reduced and, therefore, this pollutant is excluded (Criterion 6).

Chromium. Data acquired during this study measured total chromium levels (as opposed to dissolved) regardless of valence state. Extensive literature references are available for treatment of wastewater with respect to either trivalent or hexavalent chromium. However, these references predominantely address waste streams from the electroplating industry, dyes, inorganic pigments, and metal cleaning operations. The natural mineral, chromite ($FeCr_2O\propto$), has chromium in the trivalent form. Of 75 treated effluents for which chromium measurements were made, only 26 had detectable total chromium concentrations with a median value of 0.035 mg/l (for detected values only). Trivalent chromium is effectively removed by the BPT treatment, lime precipitation and settling at the pH range normally encountered in treatment systems (pH 8 to 9) associated with this industry. Consequently, it has been concluded that the concentration levels present at most facilities would not be effectively reduced further by the known technologies (Criterion 6).

Pollutants Detected in Treated Effluents at a Small Number of Facilities and Uniquely Related to Those Facilities

The toxic organic pollutant, 2,4-dimethylphenol, was detected in the effluent at only one facility (9202) during the screen sampling program. AEROFLOAT™, used as a flotation agent in ore beneficiation at this facility, is a precursor of 2,4-dimethylphenol. However, since the compound was identified only in one facility, it is excluded under Paragraph 8(1)(iii) of the Revised Settlement Agreement.

EXCLUSION OF TOXIC POLLUTANTS BY SUBDIVISION AND MILL PROCESS

The toxic parameters which did not qualify for exclusion throughout the entire category (i.e., toxic metals, cyanide, and asbestos) were evaluated for potential exclusion in each subcategory. Table VI-1 summarizes the sampling data for the toxic metals, asbestos and cyanide, by subcategory, subdivision and mill process. The number of representative samples taken and a summary of influent and effluent data are provided in the table. This table was reviewed and the data evaluated for possible exclusion from regulation based on the criteria previously discussed. In particular, Criterion 3 (not detected in treated effluents by approved analytical methods) and Criterion 6 (present at levels too low to be effectively reduced by known technologies) were used to exclude certain toxic metals and cyanide from particular subdivisions and mill processes. The toxic pollutant parameters chosen for exclusion and the exclusion criteria are summarized by category, subdivision, and mill process in Table VII-3.

CONVENTIONAL POLLUTANT PARAMETERS

Total Suspended Solids (TSS)

Total suspended solids (or suspended solids) are regulated for all subcategories under BPT effluent limitations. High suspended solid concentrations result as part of the mining process, and by crushing, grinding, and other processes commonly used in milling. Dredging and gravity separation processes also produce high suspended solids. Effluent limitations are proposed for total suspended solids under BCT.

pH

This parameter is regulated for every subcategory under BPT effluent guidelines; BCT effluent limitations will apply in the same manner. Acid conditions prevalent in the ore mining and dressing industry may result from the oxidation of sulfides in mine waters or discharge from acid leach milling processes. Alkaline-leach milling processes also contribute waste loading and can adversely affect receiving water pH.

BOD, Oil and Grease

These conventional parameters are not regulated under BPT effluent guidelines and were not found in significant concentrations during development of the data base. They are not applicable to an industry which deals primarily in inorganic substances.

NON-CONVENTIONAL POLLUTANT PARAMETERS

Settleable Solids

Solids in suspension that will settle in one hour under quiescent conditions because of gravity are settleable solids. This parameter is most useful as an indicator of the operating efficiency of sedimentation technologies, particularly sedimentation ponds, and is recommended for use as such to establish effluent limitations for gold placer mines.

Iron

Iron is very common in natural waters and is derived from common iron minerals in the substrata. The iron may occur in two forms: inherently increases iron levels present in process and mine drainage. The aluminum ore mining industry also contributes elevated iron levels through mine drainage. Iron, both total and dissolved, is regulated for segments of the industry under BPT effluent limitations and effluent guidelines are developed for iron under BAT effluent limitations.

Radium 226

Radium 226 is a member of the uranium decay series and, as discussed in Section III, it is always found with uranium ore. As a result of its long half-life (1,620 years), radium 226 may persist in the biosphere for many years after its introduction through effluents or wastes. Therefore, because of its radiological consequences, concentrations of this radionuclide must be restricted to minimize potential exposure to humans. It is regulated under BPT effluent limitations because of the radiological consequences and because data indicate that control of radium 226 also serves as a surrogate control for other radionuclii and is regulated under BAT effluent limitations.

Ammonia

Ammonia compounds (e.g., ammonium hydroxide) may be used as precipitation reagents in alkaline leaching circuits in uranium mills. The sodium diuranate which results from leaching, recarbonization and precipitation is generally redissolved in sulfuric acid to remove sufficient sodium to meet the specifications of American uranium processors. The uranium values are precipitated with ammonia to yield a yellowcake low in sodium. By-product ammonium sulfate and excess ammonia remaining may flow to wastewater treatment downstream. Consequently, ammonia is regulated under BPT and will be regulated under BAT.

CONVENTIONAL AND NON-CONVENTIONAL PARAMETERS SELECTED

A review of the data collected subsequent to BPT effluent guidelines development serves to confirm parameter selections made for BPT. No new parameters were discovered in significant quantitities in any subcategory. Therefore, development of BAT regulations for conventional and non-conventional parameters will be for the same parameters regulated under BPT. Table VII-2 illustrates the parameters to be regulated by subcategory and subpart.

SURROGATE/INDICATOR RELATIONSHIPS

The Agency believes that it may not always be feasible to directly limit each toxic which is present in a waste stream. Surrogate/indicator relationships provide an alternative to direct limitation of toxic pollutants. A surrogate relationship occurs between a toxic pollutant and a set of commonly regulated parameters when the concentration(s) of the regulated parameter(s) are used to predict the concentration of the toxic pollutant. When the concentration(s) of the regulated parameter(s) are used to predict whether or not the toxic pollutant level will be reduced, it is an indicator relationship. In the first instance, the regulated parameter(s) are called surrogates and in the second, they are called indicators.

The advantage of the surrogate/indicator relationship is that, by regulating certain conventional and non-conventional parameters, toxic pollutants are controlled to the same degree as if they had been directly controlled. Only those toxics whose concentrations can be quantitatively predicted based on knowledge of the concentration of one or more regulated parameters can be indirectly limited in this manner. Surrogates and indicators are discussed more fully in the Federal Register, Vol. 44, No. 166, pp. 34397-9.

Statistical Methods

Surrogate/indicator relationships were developed for several of the priority pollutant metals which were selected for regulation. The statistical methodology used in the development of these relationships included the following phases:

1. exploratory data analysis
2. model estimation
3. model verification

The objective of the exploratory data analysis phase was to assess the likelihood of accurately specifying the chemical and physical relationships between the priority pollutant metals and the potential surrogate/indicator parameters, given the limitations of the data available for the analysis. Summary statistics, plots, and correlations were examined. The model estimation phase quantified the relationships which were identified during the exploratory data analysis phase by using regression analysis. The model verification phase assessed the validity of the models by applying them in a simulated regulatory situation. The relationships were tested on a separate set of data from that used in the estimation phase.

Relationships

A statistical analysis of pollutant concentrations in ore mining wastewaters indicates a relationship between TSS and the following toxic pollutants:

1. chromium
2. copper
3. lead
4. nickel
5. selenium
6. zinc
7. asbestos

Therefore, when treatment technologies are employed for reducing TSS there is a reduction in the levels of these toxics. The relationship and indicated control was used in selecting technologies considered for BAT, technologies which reduce TSS, as discussed further in Section VIII.

Additonal Paragraph 8 Exclusion

As discussed in Section X, additional paragraph 8 exclusions were made during the selection of BAT options and BAT effluent limitations. These exclusions included the decision to regulate asbestos (chrysotile) by limiting the discharge of TSS as discussed in Section X. The reader is referred to the additional information and supporting data found in a separate report entitled, "Development of Surrogate/Indicator Relations in the Ore Mining and Dressing Point Source Category." Also, cyanide is not regulated because the Agency cannot quantify a reduction in total cyanide by use of any technology known to the Administrator. The Agency concluded that limitations on copper, lead, and zinc would ensure adequate control of arsenic and nickel. Finally, EPA excluded uranium mills from BAT because the pollutants found in the discharge are uniquely related to a single sources.

Table VII-1

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
ACENAPHTHENE	33	0					28	0				
ACROLEIN	33	0					28	0				
ACRYLONITRILE	33	0					28	0				
BENZENE	33	10	4.8922	4	10	10	28	3	8.3333	7	10.7	11
BENZIDENE	33	0					28	0				
CARBON TETRACHLORIDE	33	1	1	1	1	1	28	0				
CHLOROBENZENE	33	0					28	1	0.005	0.005	0.005	0.005
1,2,3-TRICHLOROBENZENE	33	0					28	0				
HEXACHLOROBENZENE	33	0					28	0				
1,2-DICHLOROETHANE	33	0					29	0				
1,1,1-TRICHLOROETHANE	33	9	6.7208	6.5811	10	10	28	5	7.2849	6.5811	10	10
HEXACHLOROETHANE	33	0					28	0				
1,1-DICHLOROETHANE	33	0					28	0				
1,1,2-TRICHLOROETHANE	33	0					28	0				
1,1,2,2-TETRACHLOROETHAN	33	0					28	0				
CHLOROETHANE	33	0					28	0				
BIS(CHLOROMETHYL) ETHER	33	0					28	0				
BIS(2-CHLOROETHYL) ETHER	33	0					28	0				
2-CHLOROETHYL VINYL ETHE	32	0					28	0				
2-CHLORONAPHTHALENE	32	0					28	0				
2,4,6-TRICHLOROPHENOL	32	1	11.667	11.667	11.667	11.667	28	0				
PARACHLOROMETA CRESOL	32	0					28	0				
CHLOROFORM	32	9	7.6096	3.1623	12.5	35	28	8	5.1281	3.1623	10	10
2-CHLOROPHENOL	32	0					28	0				
1,2-DICHLOROBENZENE	32	0					28	0				
1,3-DICHLOROBENZENE	32	0					28	0				
1,4-DICHLOROBENZENE	32	0					28	0				
3,3-DICHLOROBENZIDINE	32	0					28	0				

Table VII-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
			DETECTED VALUES ONLY				*			DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MED	90%	MAX
1,1-DICHLOROETHYLENE	32	2	6.5811	3.1623	8.6325	10	*	28	0				
1,2-TRANS-DICHLOROETHYLE	32	0					*	28	1	270	270	270	270
2,4-DICHLOROPHENOL	32	1	10	10	10	10	*	28	0				
1,2-DICHLOROPROPANE	32	0					*	28	0				
1,3-DICHLOROPROPENE	32	0					*	28	0				
2,4-DIMETHYLPHENOL	32	1	140	140	140	140	*	28	1	270	270	270	270
2,4-DINITROTOLUENE	32	0					*	28	0				
2,6-DINITROTOLUENE	32	0					*	28	0				
1,2-DIPHENYLHYDRAZINE	32	0					*	28	0				
ETHYLBENZENE	32	4	6.7167	1	13.48	17.687	*	28	3	6.6	4.9	9.64	10
FLUORANTHENE	32	0					*	28	0				
METHYL CHLORIDE	33	1	45	45	45	45	*	28	0				
METHYL BROMIDE	33	0					*	28	0				
BROMOFORM	33	0					*	28	0				
DICHLOROBROMOMETHANE	33	0					*	28	2	6.5811	3.1623	8.6325	10
TRICHLOROFLUOROMETHANE	33	5	5.0325	2.0811	10	10	*	28	3	4.7208	2.0811	7.9487	10
DICHLORODIFLUOROMETHANE	33	0					*	28	0				
CHLORODIBROMOMETHANE	33	0					*	28	0				
HEXACHLOROBUTADIENE	33	0					*	28	0				
HEXACHLOROCYCLOPENTADIEN	33	0					*	28	0				
ISOPHORONE	33	0					*	28	0				
NAPHTHALENE	33	1	12.5	12.5	12.5	12.5	*	28	0				
NITROBENZENE	33	0					*	28	0				
2-NITROPHENOL	33	0					*	28	0				
4-NITROPHENOL	33	0					*	28	0				
2,4-DINITROPHENOL	33	0					*	28	0				
4,6-DINITRO-O-CRESOL	33	0					*	28	0				

Table VII-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
N-NITROSODIMETHYLAMINE	33	0					*	28	0				
N-NITROSODIPHENYLAMINE	33	0					*	28	0				
N-NITROSODI-N-PROPYLAMIN	33	0					*	28	0				
PENTACHLOROPHENOL	33	1	10	10	10	10	*	28	0				
PHENOL	33	2	118	76	143.2	160	*	28	3	92.3	33.45	166.8	210
BIS(2-ETHYLHEXYL) PHTHAL	33	15	20.16	13	39.833	100	*	28	18	12.458	10	26	50
BUTYL BENZYL PHTHALATE	33	2	10.75	0.5	16.9	21	*	28	4	27.791	10	52.4	66
DI-N-BUTYL PHTHALATE	33	13	16.489	10	28.1	56	*	28	12	25.864	10	39.2	140
DI-N-OCTYL PHTHALATE	10	3	10	10	10	10	*	7	3	12.167	10	14.55	16.5
DIETHYL PHTHALATE	33	16	24.414	10	59.4	90	*	28	4	7.875	9.8	10	10
DIMETHYL PHTHALATE	33	0					*	28	3	12.2	5.8	20.35	25
BENZO(A)ANTHRACENE	33	0					*	28	0				
BENZO(A)PYRENE	33	0					*	28	0				
BENZO(B)FLUORANTHENE	33	0					*	28	0				
BENZO(K)FLUORANTHENE	33	0					*	28	0				
CHRYSENE	33	0					*	28	0				
ACENAPHTHYLENE	33	0					*	28	0				
ANTHRACENE	33	0					*	28	0				
BENZO(G,H,I)PERYLENE	33	0					*	28	0				
FLUORENE	33	1	10	10	10	10	*	28	1	10	10	10	10
PHENANTHRENE	33	0					*	28	0				
DIBENZO(A,H)ANTHRACENE	33	0					*	28	0				
INDENO(1,2,3-C,D)PYRENE	33	0					*	28	0				
PYRENE	33	0					*	28	0				
TETRACHLOROETHYLENE	33	2	7.75	4.5	9.7	11	*	28	1	1.1	1.1	1.1	1.1
TOLUENE	33	9	399.28	2.0811	388.3	3580	*	28	6	2.5667	1	5.28	10
TRICHLOROETHYLENE	33	0					*	28	0				

Table VII-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)						*	TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
VINYL CHLORIDE	33	0					*	28	0				
ALDRIN	33	4	6.4156	5	9	10	*	28	2	6.5811	3.1623	8.6325	10
DIELDRIN	33	0					*	28	2	6.5811	3.1623	8.6325	10
CHLORDANE	33	0					*	28	0				
4,4-DDT	33	0					*	28	0				
4,4-DDE	33	1	5	5	5	5	*	28	0				
4,4-DDD	33	1	6.6667	6.6667	6.6667	6.6667	*	28	0				
ENDOSULFAN-ALPHA	33	1	10	10	10	10	*	28	0				
ENDOSULFAN-BETA	33	0					*	28	0				
ENDOSULFAN SULFATE	33	0					*	28	0				
ENDRIN	33	0					*	28	1	5	5	5	5
ENDRIN ALDEHYDE	33	0					*	28	0				
HEPTACHLOR	33	1	7.5	7.5	7.5	7.5	*	28	2	6.5811	3.1623	8.6325	10
HEPTACHLOR EPOXIDE	33	0					*	28	0				
BHC-ALPHA	33	5	5.2649	4.0811	7.5	10	*	28	3	5	5	5	5
BHC-BETA	33	5	6.1325	5	8.75	10	*	28	1	5	5	5	5
BHC (LINDANE)-GAMMA	33	4	6.2072	5	8.6667	10	*	28	0				
BHC-DELTA	33	2	5	5	5	5	*	28	2	5	5	5	5
PCB-1242 (AROCHLOR 1242)	33	0					*	28	0				
PCB-1254 (AROCHLOR 1254)	33	0					*	28	0				
PCB-1221 (AROCHLOR 1221)	9	0					*	6	0				
PCB-1232 (AROCHLOR 1232)	9	0					*	6	0				
PCB-1248 (AROCHLOR 1248)	9	0					*	6	0				
PCB-1260 (AROCHLOR 1260)	9	0					*	6	0				
PCB-1016 (AROCHLOR 1016)	9	0					*	6	0				
TOXAPHENE	32	0					*	27	0				
2,3,7,8-TETRACHLORODIBEN	33	0					*	28	0				

Table VII-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW(UG/L)							TREATED (UG/L)					
	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX	*	NUMBER OF SAMPLES	NUMBER DETECTED	DETECTED VALUES ONLY MEAN	MED	90%	MAX
CIS 1-3-DICHLOROPROPYLEN							*						
TRAN 1,3-DICHLOROPROPYLE							*						

Table VII-1 (Continued)

DATA SUMMARY
ORE MINING DATA
ALL SUBCATEGORIES

	RAW (MG/L)						TREATED (MG/L)					
			DETECTED VALUES ONLY						DETECTED VALUES ONLY			
	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX	NUMBER OF SAMPLES	NUMBER DETECTED	MEAN	MEDIAN	90%	MAX
ANTIMONY (TOTAL)	82	6	.04333	0.028	0.1	0.1	71	3	0.034	100E-5	0.1	0.1
ARSENIC (TOTAL)	114	106	2.2219	0.3	8.43	12	100	83	.15349	0.018	0.6	1.5
BERYLLIUM (TOTAL)	84	43	.13735	0.1	0.2782	0.92	73	10	0.0051	0.005	0.0109	0.011
CADMIUM (TOTAL)	196	54	0.213	0.0225	0.655	1.2	92	36	.01415	0.005	0.06	0.077
CHROMIUM (TOTAL)	85	70	3.9162	0.81	11	18	75	26	.13623	0.035	0.332	1.8
COPPER (TOTAL)	163	100	65.221	5.8	224.5	464	90	83	.23464	0.06	0.548	4.6
CYANIDE (TOTAL)	65	24	.32042	0.21	0.93	1.24	57	14	.13571	0.08	0.435	0.6
LEAD (TOTAL)	66	70	4.1907	1.35	4.98	130	75	31	.13481	0.05	0.376	0.959
MERCURY (TOTAL)	87	54	.06399	.00145	0.013	0.02	69	37	.01264	800E-6	0.0306	0.25
NICKEL (TOTAL)	86	70	3.4166	2.5	9.3	14.2	75	43	.22202	0.07	0.966	1.28
SELENIUM (TOTAL)	84	58	0.2309	0.15	0.615	1.5	73	37	.05557	0.015	0.112	0.9
SILVER (TOTAL)	84	24	.34624	0.21	0.77	1.1	73	9	.01567	0.019	0.04	0.04
THALLIUM (TOTAL)	82	3	.26433	1.17	1.24	1.24	71	3	.52767	0.74	0.84	0.84
ZINC (TOTAL)	196	196	33.885	0.875	292.4	300	92	82	.90209	0.062	2.244	11.1
COD	22	22	302.73	10.95	1761	1900	23	23	11.698	11	20.6	53
TSS	47	47	3156.8	40	836	95450	47	46	24.989	9.5	69.2	157
TOC	6	6	485	560	750	750	9	8	5.1875	5.25	6	6
PH (UNITS)	35	35	6.3934	7	8.44	9.5	36	36	6.9714	7.7	8.16	8.5
PHENOLICS (4AAP)	72	71	.11372	0.04	0.3422	0.75	57	49	.07378	0.032	0.21	0.46
IRON (TOTAL)	22	19	523.7	6.208	1990	2040	25	21	.62619	0.209	2.192	3.87

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TABLE VII-2. POLLUTANTS CONSIDERED FOR REGULATION

Subcategory	Sub-Division	Mill Process	Toxic Pollutants																Conventionals		Nonconventionals					
			Sb	As	Chrysotile	Be	Cd	Cr	Cu	CN	Pb	Hg	Ni	Se	Ag	Tl	Zn	Total Phenolics	TSS	pH	COD	Fe – Total or Dis.	Ra 226	Al	U	Settleable Solids
Iron Ore	Mines		E	E	G	E	E	E	E	E	E	E	E	E	E	E	E	E	G	G		G				
	Mills	Phys/Chem	E	E	G	E	E	E	E	E	E	E	E	E	E	E	E	E	G	G		G				
		Phys (Mesabi)	Zero Discharge at BPT																							
Copper, Lead, Zinc, Gold, Silver, Platinum, Molybdenum	Mines		E	E	G	E	G	E	G	G	G	G	G	E	E	E	G	E	G	G						
	Mills	Cyanidation r Amalgamation	Zero Discharge at BPT																							
		Heap, Vat, Dump, In Situ Leach	Zero Discharge at BPT																							
		Froth Flotation	E	G	G	E	G	E	G	G	G	G	G	E	E	E	G	G	G	G						
		Gravity Separation																		G						G
Aluminum	Mines		E	E	G	E	E	E	E	E	E	E	E	E	E	E	E	E	G	G		G		G		
Tungsten	Mines		E	G	G	E	G	E	G	E	E	E	E	E	E	E	G	E	G	G						
	Mills		E	G	G	E	G	E	G	E	G	E	E	E	E	E	G	E	G	G						
Mercury	Mines											G	G						G	G						
	Mills		Zero Discharge at BPT																							
Uranium	Mines		E	E	G	E	E	E	E	E	E	E	E	E	E	E	G	E	G	G	G		G		G	
	Mills, In Situ Leach		E	E	G	E	E	E	E	E	E	E	G	E	E	E	G	E	G	G	G		G			
Antimony	Mines		Reserved																							
	Mills																									
Titanium	Mines		E	E	G	E	E	E	E	E	E	E	E	E	E	E	E	E	G	G		G				
	Mills		E	E	G	E	E	E	E	E	E	E	G	E	E	E	G	E	G	G						
	Mills w/ Dredges		E	E	G	E	E	E	E	E	E	E	E	E	E	E	E	E	G	G		G				
Nickel	Mines		Reserved																							
	Mills																									
Vanadium	Mines		Reserved																							
	Mills																									

E = Excluded from Guideline Development
G = Guidelines to be Considered

TABLE VII-3. PRIORITY METALS EXCLUSION CRITERIA BY SUBCATEGORY, SUBDIVISION, MILL PROCESS

Subcategory	Sub-Division	Mill Process	Toxic Pollutants															
			Sb	As	Chrysotile	Be	Cd	Cr	Cu	CN	Pb	Hg	Ni	Se	Ag	Tl	Zn	Total Phenolics
Iron Ore	Mines		3	6		3	3	3	6	3	3	3	3	3	3	3	6	3
	Mills	Phys/Chem	3	6		3	3	6	6	3	3	3	3	3	3	3	6	3
		Phys (Mesabi)	Zero Discharge at BPT															
Copper, Lead, Zinc, Gold, Silver, Platinum, Molybdenum	Mines		3	6		6		3						3	6	6		6
	Mills	Cyanidation of Amalgamation	Zero Discharge at BPT															
		Heap, Vat, Dump, In Situ Leach.	Zero Discharge at BPT															
		Froth Flotation	6			6		6						6	6	3		6
		Gravity Separation																
Aluminum	Mines		3	3		3	3	6	6	3	3	6	3	3	3	3	3	6
Tungsten	Mines		3			3				3	3	6	3	3	3	3		3
	Mills		6			6				3		3	3	3	6	3		6
Mercury	Mines																	
	Mills		Zero Discharge at BPT															
Uranium	Mines		3	6		3	3	6	3	3	6	6	3	6	3	3		6
	Mills, In Situ Leach		3			3	3	6	6	3	6	6		6	6	3		6
Antimony	Mines		Reserved															
	Mills																	
Titanium	Mines		3	3		3	3	3	6	3	6	3	3	3	3	3	6	6
	Mills		6	3		3	3	3	6	3	6	3		3	3	3		6
	Mills w/ Dredges		3	3		3	6	3	6	3	6	6	3	3	6	3	6	6
Nickel	Mines		Reserved															
	Mills																	
Vanadium	Mines		Reserved															
	Mills																	

EXCLUSION CRITERIA

1. Equal or more stringent protection is already provided by EPA's guidelines.
2. The pollutant is present as a result of its presence in the intake water.
3. The pollutant is not detected by approved methods.
4. The pollutant is unique to a small number of sources.
5. The pollutant is present in trace, non-toxic amounts.
6. The pollutant is present in amounts too small to treat.
7. The pollutant is effectively controlled by treating other pollutants.

Table VII-4

TUBING LEACHING ANAYSIS RESULTS

Component	Micrograms/Liter Original	Tygon
Bis (2-ethylhexyl) Phthalate		
Acid Extract	915	N.D.
Base-Neutral Extract	2,070	885
Phenol		
Acid Extract	19,650	N.D.
Base-Neutral Extract	N.D.	N.D.

N.D. - Not Detected

SECTION VIII

CONTROL AND TREATMENT TECHNOLOGY

This section discusses the techniques for pollution abatement applicable to the ore mining and milling industry. General categories of techniques are: in-process control, end-of-pipe treatment, and best management practices. The current or potential use of each technology in this and similar industries and the effectiveness of each are discussed.

Selection of the optimal control and treatment technology for wastewater generated by this industry is influenced by several factors:

1. Large volumes of mine water and mill wastewater must be controlled and treated. In the case of mine water, the operator often has little control over the volume of water generated except for diversion of runoff from surface mine areas.

2. Seasonal and daily variations in the amount and characteristics of mine water are influenced by precipitation, runoff, and underground water contributions.

3. There are differences in wastewater composition and treatability caused by ore mineralogy, processing techniques, and reagents used in the mill process.

4. Geographic location, topography, and climatic conditions often influence the amount of water to be handled, treatment and control strategies, and economics.

5. Pilot plant testing and acquisition of empirical data may be necessary to determine appropriate treatment technologies for the specific site.

6. The availability of energy, equipment, and time to install the equipment must be considered. Selection of BAT by mid-1980 will give the industry three years to implement the technology.

IN-PROCESS CONTROL TECHNOLOGY

This section discusses process changes available to existing mills to improve the quality or reduce the quantity of wastewater discharged from mills. The techniques are process changes within existing mills.

Control of Cyanide

Cyanide is a commonly used mill process reagent, used in froth flotation as a depressant and in cyanidation for leaching.

Froth Flotation

In the flotation of complex metal ores, depressing agents assist in the separation of one mineral from another when flotabilities of the two minerals are similar for any given combination of flotation reagents. Cyanide is a widely used depressant, either in the form of crude calcium cyanide flake or sodium cyanide solution. Alkaline cyanides are strong depressants for the iron sulfides (pyrite, pyrrhotite, and marcasite), arsenopyrite, and sphalerite. They also act as depressants, to a lesser extent, for chalcopyrite, enargite, tannantite, bornite, and most other sulfide minerals, with the exception of galena (Reference 1). Cyanide, in some instances, cleans tarnished mineral surfaces, thereby allowing more selective separation of the individual minerals (Reference 2).

In flotation, cyanide has primarily been used to aid in the separation of galena from sphalerite and pyrite. It also has been used to separate silver and copper sulfides from pyrite, nickel and cobalt sulfides from copper sulfides, and molybdenum sulfide from copper sulfide.

In beneficiation of base metal ores by flotation, the rate of cyanide addition to the circuit must be varied to optimize both the percentage recovery and concentrate grade of the various metals recovered (References 1, 2, 3, and 4). The addition of either too much or too little cyanide can result in loss of recovery and reduction in the grade of concentrate. For example, in selective flotation of copper, lead/zinc, and copper/lead/zinc ores, the addition of too little cyanide will result in the flotation of pyrite, thereby reducing the copper, lead, and zinc concentrate grades. Also, too little cyanide will result in flotation of zinc in the lead circuit, which produces a lower lead concentrate grade.

Cyanide control is also desirable from a waste treatment standpoint. Excess cyanide use subsequently requires more copper sulfate when zinc is activated for flotation. This not only represents uneconomical use of reagents, but also increases the waste loading of both copper and cyanide. Reagent use at various domestic base metal flotation mills and comments relative to the efficiency of cyanide use in these mills are described in Section VI, Summary of Reagent Use in Flotation Mills.

Many mills have replaced valve operated reagent feeders for cyanide addition with metered feeders, such as the Clarkson or Geary feeder, which maintain constant flow of a controlled solution of cyanide. The use of these metered feeders influence the amount of cyanide fed to the process by insuring that the proper amount required is added and, thereby, reducing the possibility of "overshooting" the correct dosage. Also, some of these same mills have imposed restrictions on which personnel can adjust these automatic feeders to eliminate the arbitrary increase in dosage that can overshoot the minimum amount required to produce the most efficient separation.

The degree of sophistication of in-process control of cyanide varies widely in the category. The greatest degree of sophistication is used at copper/lead/zinc Mill 3103. A Courier online X-ray analyzer performs analyses at 10-minute intervals of the mill heads and tails and of the concentrates, heads, and tails of the individual flotation circuits. Analytical results are computed by a Honeywell 316 computer and automatically printed and charted. The mill operator may then adjust the rate of reagent addition based on these analytical results. For example, the rate of cyanide addition is decreased when the copper content of the copper circuit tailings increases during a time increment (usually two hours). Conversely, the rate of cyanide addition is increased when the iron content of the lead and zinc concentrates increases. In this manner, the mill operator is able to optimize the reagent use, percentage recovery, and grade of concentrate produced. Several mills (3103, 3105, 3122, 3123, 2117, and 2121) have on-stream analytical capabilities.

Laboratory analysis provides adequate control in the milling of simple, "clean" ores. However, the greater the complexity and variability of the ore being milled, the more advantageous it becomes to a mill operator to have on-stream analytical capabilities.

The prevailing practice for in-process control of reagent addition consists of manual sampling and laboratory analysis of heads, tails, and concentrates. Typically, samples are collected at two-hour intervals and analyses are begun immediately. Approximately two hours are required before analytical results are available. This method is slower and produces less information than the more sophisticated method previously described.

Many small mills have limited analytical capabilities and the control of reagent addition depends on the experience of the mill operator. According to mill operators and site visit data, cyanide addition in excess of the amount required is generally used with limited analytical control.

Control of the rate of reagent addition depends on the attention given to the analytical results by the mill operator. The attitude, conscientiousness, and experience of the mill operator have a significant effect on the degree of control maintained over reagent usage. The efficiency of reagent usage impacts the overall efficiency and economy of the mill, as well as the character of the wastewater generated, and operators must remain aware of this.

Cyanidation

Cyanide is also used prominently in processing lode gold and silver ores by a leaching process (the cyanidation process) which uses dilute, weakly alkaline solutions of potassium or sodium cyanide. In-process control of cyanide at cyanidation mills involves recycle of the spent leach solutions. This control practice is, therefore, beneficial in two respects. First, the cyanide wasteload is greatly reduced, making treatment more economical. Second, since a fraction of the cyanide is recovered for reuse, the cost of reagent is reduced. The BPT effluent guidelines for cyanidation mills is no discharge of process wastewater.

Alternatives to Cyanide In Flotation

Cyanide is believed to function primarily as a reducing agent in the depression of pyrite in xanthate flotation operations. In 1970, Miller (Reference 5) investigated alternative reducing agents and found that, in terms of effectiveness and cost, sodium sulfite compared quite favorably with cyanide as a pyrite depressant. In particular, it was found that cyanide exerted some depressant effect on chalcopyrite and pyrite, but sodium sulfite did not. The sodium sulfite alternative appeared to be applicable to copper ore flotation operations.

Some mills use sulfite or sulfides instead of cyanide. Mill 3101 is an example of a copper/zinc flotation mill which uses sodium sulfite and no cyanide. There was no measurable effect on recovery or grade of concentrate. At Mill 6104, copper and molybdenum minerals are separated in froth flotation with sodium bisulfide used as a copper depressant, provide another alternative to the use of cyanide for the depression of copper minerals in selective flotation.

An EPA-sponsored study to identify and evaluate alternatives to sodium cyanide was initiated in May 1978 (References 6 and 7) and alternatives were identified by a literature search. Points taken into consideration were: ability to depress pyrite, selectivity of depressant, theory of performance, inferred and specific environmental aspects, state of development as a practical depressant, and cost. Fourteen alternatives were identified, three of which were carried into the evaluation phase of the study. The compounds selected for bench-scale evaluation were: sodium monosulfide (Na_2S), sodium sulfite (Na_2SO_3), and sodium thiosulfate ($Na_2S_2O_3$). Three types of ore (copper, copper/lead/zinc, and zinc) were chosen for the flotation experiments. All of the ores contained pyrite.

The results of this study are summarized in Table VIII-1. The most effective depressant in the copper ore experiments was sodium cyanide. Sodium sulfite at 0.504 kg/metric ton (1.008 pound/short ton) of ground ore approached the effectiveness of the cyanide at the natural pH level, natural meaning the prevailing pH of the ground ore plus water. At elevated pH (10 to 12), sodium sulfite and sodium monosulfide surpassed cyanide in the amount of copper recovered, but these were less effective in depressing the pyrite.

When dealing with copper/lead/zinc ore, it is desirable to float the copper and lead initially, while depressing the iron and zinc. At the natural pH level, the sodium sulfite equaled the cyanide in recovery of copper and lead and was superior to the cyanide in depressing iron and zinc. At pHs of 10 to 12, sodium sulfite surpassed the cyanide in the recovery of copper and lead and nearly equaled the cyanide in depression of iron and zinc. Sodium monosulfide resulted in good recoveries of copper and lead, but not as good as other alternatives. It was ineffective in depressing the pyrite.

In the experiments with the zinc ore, only sodium sulfite and sodium monosulfide were studied. Zinc/pyrite ore is one of the most difficult ores to float and the study confirms this. Techniques used to improve the floatability of zinc ore were not applied in the experiments. At the natural pH level, the low level of sodium sulfite surpassed sodium cyanide in the recovery of zinc and was slightly less effective in the suppression of iron. At elevated pH values, all of the alternatives studied, including the absence of a depressant, out-performed sodium cyanide. Sodium monosulfide was the most effective alternative under the high pH conditions.

In summary, bench-scale tests indicate that sodium sulfite is a potential substitute for sodium cyanide. Also, sodium monosulfide is fairly effective at high pH. However, these are bench scale tests, and full-scale operations in this industry rarely equate directly to bench-scale results. Typically, extensive bench and pilot scale testing with the particular ore to be milled are conducted by an operator before the decision to convert is made. Even then, weeks or months of adjustments may be necessary to optimize the new process.

Reagent cost estimates are given in Table VIII-1, and the difference in cost is negligible. However, reagent cost is only one of the economic considerations. Components of the cost are (1) reagent costs, (2) downtime, (3) laboratory process simulation costs, (4) equipment cost, and (5) optimization costs. The last are probably the highest. Interviews with operators revealed that downtime may be only a few days, but optimizing the process may take a year and concentration grades, they fear, would never reach current standards. The financial penalties can be severe, as evidenced by one mill's report on smelter penalties for offgrade lead concentrates (mill process 700 TPD, Reference 7):

1. For every 0.1 percent of copper in excess of 1.0 percent, the penalty could amount to $96,000 per year.

2. For every 0.1 percent of iron in excess of 4 percent, the penalty could amount to $264,000 per year.

The study concludes that conversion costs are complex and cannot be accurately estimated (Reference 7).

Therefore, cyanide substitution should not be the basis for selection of BAT effluent guidelines, as the cost of substitution cannot be calculated and an economic analysis cannot be conducted. However, if a particular mill can meet BAT effluent guidelines by reagent substitution and maintain concentrate quality, that option is available.

Alternatives to Use of Phenolic Compounds As Mill Reagents

Several phenolic compounds are used in this industry. The most common is cresylic acid, which is essentially 100 percent phenolics, and is used as a frothing agent at several base and precious metals flotation mills (e.g., 2117 and 4403). A frother, pine oil, used in sulfide mineral flotation, is composed essentially of terpene alcohols, terpene ketone, and terpene hydrocarbons. These terpene compounds are not phenolics, but some phenolics are likely to be present as byproducts of the steam distillation process used to produce them. Several collectors (promoters), such as Reco and AEROFLOAT, also contain phenolic radical groups. In isolated instances, depressants containing phenolics have been used. At one mill (2120), a phenolic compound (Nalco 8800) is used as a wetting agent for dust control during secondary ore crushing. In this latter case, nonphenolic wetting agents, including olefinic compounds and petroleum-based sulfonates, are being considered for use.

The flotation reagents and dosages used vary widely from mill to mill (refer to Table VI-19). Reagent and dosage rate selection is a complex process that often takes years to optimize and is continuously reevaluated at individual mills. Considerations include reagent cost and availability, compatibility with other reagents, effect on concentrate grade and metal recoveries, consistency of the ore body, and environmental impact of chemical residuals in the wastewater discharge. Selection of dosage rate is essentially a trial and error process of optimizing concentrate grade and metal recoveries and is dependent upon in-process control.

The chemistry of flotation is complex and reagent substitution may have repercussions throughout a circuit. However, a large number of nonphenolic frothers are promising as alternatives to phenol-based or phenol-containing compounds. Among the most popular nonphenolic frothers are methyl isobutyl carbinol (MIBC) and polyglycol methyl ethers. Frothers are generally nonselective, generically related compounds, with fairly predictable characteristics. As such, substitution within this class of reagents (frothers) should not be difficult. For example, Mill 2121 has recently discontinued use of cresylic acid in a silver flotation circuit by substitution with polyglycol methyl ethers.

Collectors are much more selective than frothers, and their effectiveness is highly dependent upon their compatibility with associated modifiers, promoters, activators, and depressants. Possible alternatives to phenolic collectors are dithiophosphate salts and dithiophosphoric acids with alkyl groups in place of phenol groups. Substitution of these reagents for phenol containing collectors may be feasible without serious complications or economic consequences; however, the consequences of substitution are site dependent and require extensive experimentation at each mill.

In-Process Recycle of Waste Streams

In-process recycle of concentrate thickener overflow and/or recycle of filtrate produced by concentrate filtering is practiced at a number of flotation mills (e.g., 2121, 3101, 3102, 3108, 3115, 3116, 3119, 3123, and 3140). In addition, several mills (2120, 6101, and 6157) use thickeners to reclaim water from tailings prior to the final discharge of these tailings to ponds. Water reclaimed in this manner is used as makeup water in the mill. In-process recycle of waste streams produced by concentrate dewatering is incorporated primarily as a process control measure for the recovery of metals which would otherwise be lost in the tailings. This latter practice is intended as a safeguard in the event of concentrate filter malfunctions, which would allow large quantities of metals to pass through the filter. To avoid this, these process waters are generally returned to the flotation circuits.

These practices conserve water and recover metals which would otherwise be in the wastewater discharge. The in-process recycle of concentrate thickener overflow and/or filtrate produced by concentrate filtering reduces the volume of wastewater discharged by 5 to 17 percent. Likewise, mills which use thickeners to reclaim water from tailings reduce both the new water requirement and the volume of wastewater discharged by 10 to 50 percent.

In-process recycle to reduce wastewater volume can improve the performance of existing treatment systems. For example, as the volume of wastewater discharged from a mill decreases, the retention time within the tailing pond increases. As a result, conditions favorable to settling of solids, formation of metal precipitates, and degradation of flotation reagents and cyanide (by chemical, physical, and biochemical mechanisms) are enhanced. Therefore, the in-process recycle of wastewater can be an effective means of improving the capabilities of existing tailing ponds.

Recycle of spilled reagent can also be an advantage. At Mill 3101, the occurrence of spills and overflow from flotation cells results from the milling of a higher grade ore than the reagent dosage is optimized for. A system has been implemented to collect spills and return them to the flotation circuit. This control practice not only improves the quality of treated wastewater, but the percentages of metals recovered as well.

Use of Mine Water as Makeup in the Mill

A large number of mine/mill operations use mine drainage as makeup in the mill (e.g., 4103, 4104, 4105, 3101, 3102, 3103, 3104, 3105, 3106, 3108, 3110, 3113, 3118, 3119, 3122, 3123, 3126, 3127, 3138, 3142, 6102, 6104, 9402, and 9445). In some instances, the entire process water requirement of the mill is obtained from mine drainage.

From a wastewater treatment aspect for facilities allowed to discharge, a great advantage is gained by this practice. First, this practice either eliminates the requirement for a mine water treatment system or greatly reduces the volume of wastewater discharged to a single system. As discussed previously, reducing the volume of wastewater flow to an existing treatment system can be an effective means of enhancing the capabilities of that system. Second, in situations where mine water contains relatively high concentrations of soluble metals, its use in the mill provides a more effective means for the removal of these metals than could generally be attained by treatment of the mine water alone. This is due to reduced metals solubility in the alkaline conditions maintained in flotation and most mill circuits. Therefore, use of mine water as makeup in a mill can be considered a control practice which improves the quality of mine and mill treated wastewater.

Techniques for Reduction of Wastewater Volume

Pollutant discharges from mining and milling sites may be reduced by limiting the total volume of discharge, as well as by reducing pollutant concentrations in the wastestream. Volumes of mine discharges are not, in general, amenable to control, except insofar as the mine water may be used as input to the milling process in place of water from other sources. Techniques for reducing discharges of mill wastewater include limiting water use, excluding incidental water from the waste stream, recycle of process water, and impoundment with water lost to evaporation or trapped in the interstitial voids in the tailings.

In most of the industry, water use should be reduced to the extent practical, because of the existing incentives for doing so (i.e., the high costs of pumping the high volumes of water required, limited water availability, and the cost of water treatment facilities). Incidental water enters the waste stream directly through precipitation and through the resulting runoff influent to tailing and settling ponds. By their very nature, the water-treatment facilities are subject to precipitation inputs which, due to large surface areas, may amount to substantial volumes of water. Runoff influxes are often many times larger, however, and may be controlled to a great extent by diversion ditches and (where appropriate) conduits. Runoff diversion exists at many sites and is under development at others.

Complete Recycle - Zero Discharge

Mill Water

Recycle of process water is currently practiced where it is necessary due to water shortage, where it is economically advantageous because of high make-up water costs, or the cost to treat and discharge. Some degree of recycle is accomplished at many ore mills, either by reclamation of water at the mill or by the return of decant water to the mill from the tailing pond or secondary impoundments. Recycle is becoming, and will continue to become, a more frequent practice. The benefits of recycle in pollution abatement are manifold and frequently are economic as well as environmental. By reducing the volume of discharge, recycle may not only reduce the gross pollutant load, but also allow the employment of abatement practices which would be uneconomical on the full waste stream. Further, by allowing concentrations to increase in some instances, the chances for recovery of certain waste components to offset treatment cost--or, even, achieve profitability--are substantially improved. In addition, costs of pretreatment of process water--and, in some instances, reagent use--may be reduced.

Recycle of mill water almost always requires some treatment of water prior to its reuse. In many instances, however, this may entail only the removal of solids in a thickener or tailing basin. This is the case for physical processing mills, where chemical water quality is of minor importance, and the practice of recycle is always technically feasible for such operations.

In flotation mills, chemical interactions play an important part in recovery, and recycled water may, in some instances, pose problems. The cause of these problems, manifested as decreased recoveries or decreased product purity, varies and is not, in general, well-known, being attributed at various sites and times to circulating reagent buildup, inorganic salts in recycled water, or reagent decomposition products. Experience in arid locations, however, has shown that such problems are rarely insurmountable. In general, plants practicing bulk flotation on sulfide ores can achieve a high degree of recycle of process waters with minimal difficulty or process modification. Complex selective flotation schemes can pose more difficulty, and a fair amount of work may be necessary to achieve high recovery with extensive recycle in some circuits. Problems of achieving successful recycle operation in such a mill may be substantially alleviated by the recycle of specific process streams within the mill, thus minimizing reagent crossover and degradation. The flotation of non-sulfide ores (such as scheelite) and various oxide ores using fatty acids, etc., has been found to be quite sensitive to input water quality. Water recycle in such operations may require a high degree of treatment of recycle water. In many cases, economic advantage may still exist over treatment to levels which are acceptable for discharge, and examples exist in current practice where little or no treatment of recycle water has been required.

A large number of active mills employ recycle of process wastewater and achieve zero discharge. The following list is not all inclusive, but serves to illustrate the degree to which this practice has been adopted in the ore milling industry:

Iron Mills		Copper Mills	
1101	1118	2103	2118
1102	1122	2108	2139
1103	1123	2109	2140
1105	1129	2113	2141
1106	1138	2115	2146
1112		2116	2147

Lead/Zinc		Mercury
3105	2126	9202
3123	2143	

Copper ore leaching (heap, dump, in-situ) operations practice recycle in order to reuse the acid and to maximize the extraction of copper values by hydrometallurgical methods.

Technical limitations on recycle in other ore leaching operations center on inorganic salts. The deliberate solubilization of ore components, most of which are not to be recovered, under recycle operations can lead to rapid buildup of salt loads incompatible with subsequent recovery steps (such as solvent extraction or ion exchange). In addition, problems of corrosion or scaling and fouling may become unmanageable at some points in the process. The use of scrubbers for air-pollution control on roasting ovens provides another substantial source of water where recycle is limited. At leaching mills, roasting will be practiced to increase solubility of the product material. Dusts and fumes from the roasting ovens may be expected to contain appreciable quantities of soluble salts. The buildup of salts in recycled scrubber water may lead to plugging of spray nozzles, corrosion of equipment, and decreased removal effectivenes as salts crystallizing out of evaporating scrubber water add to particulate emissions.

Impoundment and evaporation are techniques practiced at many mining and milling operations in arid regions to reduce discharges to, or nearly to, zero. Successful employment depends on favorable climatic conditions (generally, less precipitation than evaporation, although a slight excess may be balanced by process losses and retention in tailings and product) and on availability of land consistent with process-water requirements and seasonal or storm precipitation influxes. In some instances where impoundment is not practical on the full process stream, impoundment and treatment of smaller, highly contaminated streams from specific process may afford significant advantages.

Total and partial recycle have become more common in recent years. Facilities that use recycle are often in arid regions because of the scarcity of available water. Many facilities both in arid and humid regions recycle their process wastewater.

Mine Water

Complete recycle of mine drainage is generally not a viable option because often an operator has little control over water which infiltrates the mine. Except for small amounts of water used in dust control, cooling, drilling fluids, and transport fluids for sluicing tailings back to the mine for backfill, water is not widely used in the actual mining. In some cases, mine drainage is used by the mill as process water in beneficiation. However, the volume of mine drainage may exceed the mill's requirement for process water, making complete use of mine drainage unachievable.

Other Process Changes

Mill 4105 has, as a result of environmental regulation, discontinued the use of mercury (amalgamation) for the recovery of gold. The process change used consists of incorporation of a cyanidation circuit, described as a carbon-in-pulp circuit. This process technology is described in detail in Appendix A.

At uranium Mill 9405, a process change has recently been implemented specifically as a pollution control measure. Yellowcake precipitation with sodium hydroxide, rather than ammonia, is now used to reduce ammonia levels entering the receiving stream. Although only limited experience has been gained, plant personnel have noted a reduction in product grade resulting from the process change. However, product grade is apparently still within acceptable limits.

Mill 9403 has indirectly eliminated all wastewater discharges recently by eliminating their resin-in-pulp circuit (see Section III). This change was not based solely on environmental considerations, but resulted from a variety of factors which included ore characteristics, process economics, and pollution control requirements.

END-OF-PIPE TREATMENT TECHNIQUES

This subsection presents discussions of several end-of-pipe techniques which are used in industry or are applicable to the treatment problems encountered. These technologies were considered as possible BAT technologies. However, it should be noted that at many facilities in the industry, implementation of additional technology beyond BPT will not be necessary to meet the limitations based on BAT technology. The reasons for this are facility specific and may include low-waste loading due to clean ore, extremely well managed treatment systems, existing systems exceeding BPT requirements, extensive reuse of wastewater, and water conservation practices. The description of the candidate BAT technologies includes the discussion of the processes involved and their degree of use in the industry, treatability data collected by Agency contractors, and finally, historical data where available.

Technique Description

Secondary Settling

Ponds are used in the industry for settling. Tailings ponds receive relatively high solids loading and therefore require frequent cleaning or enlargement. Primary settling ponds for mine drainage used to meet BPT effluent guidelines have larger surface areas, receive larger solids loadings than secondary ponds, and may not require cleaning or dredging. Secondary settling ponds are sometimes used to provide better solids removal by plain (nonchemical aided) sedimentation.

In theory, several ponds in a series will not remove any more solids than one large pond of equal size, since the theoretical detention time in the two situations are identical. However, many sediment ponds currently in use in this industry have not been designed, operated, and maintained so as to optimize settling efficiency. Therefore, in practice, providing secondary settling in a series of ponds has been demonstrated to provide additional reduction of suspended solids in this industry.

For example, short circuiting in the primary pond (either tailing or settling), too much depth in the primary pond, shock hydraulic loads (such as precipitation runoff), and an improper discharge structure in the primary pond are all cases where secondary settling ponds can remove significant quantities of solids by plain settling.

Coagulation and Flocculation

Coagulation and flocculation are terms often used interchangeably to describe the physiochemical process of suspended particle aggregation resulting from chemical additions to wastewater. Technically, coagulation involves the reduction of electrostatic surface charges and the formation of complex hydrous oxides. Time required for coagulation is short; only what is necessary for dispersing the chemicals in solution. Flocculation is the physical process of the aggregation of wastewater solids into particles large enough to be separated by sedimentation, flotation, or filtration. Flocculation typically requires a detention of 30 minutes.

For particles in the colloidal and fine supracolloidal size ranges (less than one to two micrometers), natural stabilizing forces (electrostatic repulsion, physical repulsion by absorbed surface water layers) predominate over the natural aggregating forces (van der Waals) and the natural mechanism which tend to cause particle contact (Brownian motion). The function of chemical coagulation of wastewater may be the removal of suspended solids by destabilization of colloids to increase settling velocity, or the removal of soluble metals by chemical precipitation or adsorption on a chemical floc.

The inorganic coagulants, or flocculants, commonly used in wastewater treatment are aluminum salts such as aluminum sulfate (alum), lime, or iron salts such as ferric chloride. Hydroxides of iron, aluminum, or (at high pH) magnesium form gelatinous flocs which are extremely effective in enmeshing fine wastewater solids. These hydroxides are formed by reaction of metal salt coagulants with hydroxyl ions from the natural alkalinity in the water or from the addition of lime or another pH modifier. Sufficient natural iron and/or magnesium is normally present in wastewater of this industry so that effective coagulation can be achieved by merely raising the pH with lime addition. Lime and metal salt coagulants also act to destabalize colloidal solids, neutralizing the negatively charged solids by adsorption of cations.

Polymeric organic coagulants, or polyelectrolytes, can be used as primary coagulants or in conjunction with lime or alum as a coagulant aid. Polymeric types function by forming physical bridges between particles, thereby causing them to agglomerate. Polymers also act as filtration aids by strengthening flocs to minimize floc shearing at high filtration rates.

Coagulants are added upstream of sedimentation ponds, clarifiers, or filter units to increase the efficiency of solids separation. This practice has also been shown to improve dissolved metals removal due to the formation of denser, rapidly settling flocs, which appear to be more effective in adsorbing and absorbing fine metal hydroxide precipitates. The major disadvantage of coagulant addition to the raw wastewater stream is the production of large quantities of sludge, which must remain in perpetual storage within tailing ponds. Coarser mineral materials thicken as particulate (nonflocculant) suspensions, yet most materials (especially pulps, precipitates, slimes, tailings, and various wastewater treatment sludges) are flocculant suspensions and behave quite differently. Sedimentation is the only process occurring during thickening of particulate suspensions, with the weight of the particles borne solely by hydraulic forces. Two physically different processes occur during thickening of flocculant suspensions: sedimentation of separate flocs and consolidation of the flocculant porous medium, in which the weight of the particles is borne partially by mechanical means and partially by hydraulic forces. In efforts to reduce the solids load on primary sedimentation units, several mine/mill wastewater treatment systems add chemical coagulants after the larger, more readily settled particles have been removed by a settling pond or other treatment. Polyelectrolyte coagulants are usually added in this manner.

In most cases, chemical coagulation can be used with minor modifications and additions to existing treatment systems, although the cost for the chemicals is often significant. However, a model coagulation and flocculation system may consist of a mixing basin, followed by a flocculation basin, followed by a clarifier or settling pond and possibly a filtration unit. The purpose of the mixing basin is to disperse the coagulant into the waste stream; the reason for the flocculation basin is to increase the collisions of coagulated solids so that they agglomerate to form settleable or filterable solids. This is accomplished by inducing velocity gradients with slowly revolving mechanical paddles or diffused air.

A low capital cost alternative to the model system and one that is well suited to the industry involves introduction of the coagulant directly into wastewater discharge lines, launders, or conditioners (in the flotation process). The coagulated wastewater is then discharged to a sedimentation pond or tailing pond to effect flocculation and sedimentation of the coagulated solids. The advantages of this system, as opposed to the model treatment facility, are minimization of treatment units and capital expenditures, and treatment simplicity resulting in reduced maintenance and increased system reliability. Disadvantages of this system are lack of control over the individual treatment processes and potentially reduced removal efficiency.

The effectiveness and performance of individual flocculating systems must be analyzed and optimized with respect to mixing time, chemical-coagulant dosage, retention time in the flocculation basin (if used) and peripheral paddle speed, settling (retention) time, thermal and wind-induced mixing, and other factors.

Coagulation and flocculation are used at several facilities in this industry. Coagulants (polymers) are presently used for wastewater treatment at Mine/Mills 4403, 3121, 3120, and 1108 and at Mine 3130. In the past, flocculants have also been employed at Mine/Mills 2121 and 3114. At Mine/Mill 1108, the tailing pond effluent is treated with alum, followed by polymer addition and secondary settling to reduce suspended solids from approximately 200 mg/l to an average of 6 mg/l. At Mine/Mill 3121, initiation of the practice of polymer addition to the tailings has greatly improved the treatment system capabilities. Concentrations of total suspended solids (TSS), lead, and zinc in the tailing-pond effluent have been reduced over concentrations previously attained, as shown in the tabulation below (company-supplied data):

Parameter	Effluent Levels (mg/l) Attained Prior to Use of Polymer		Effluent Levels (mg/l) Attained Subsequent to Use of Polymer	
	Mean	Range	Mean	Range
TSS	39	15 to 80	14	4 to 34
Pb	0.51	0.24 to 0.80	0.29	0.14 to 0.67
Zn	0.46	0.23 to 0.86	0.38	0.06 to 0.69

Similarly, the use of a polymer at Mine 3130 reduced treated effluent concentrations of total suspended solids, lead, and zinc over concentrations attained prior to use of the polymer, as shown in the following (company supplied data):

Parameter	Effluent Levels (mg/l) Attained Prior to Use of Polymer and Secondary Settling Pond*		Effluent Levels (mg/l) Attained Subsequent to Use of Polymer and Secondary Settling Pond*	
	Mean	Range	Mean	Range
TSS	19	4 to 67	2	0.02 to 6.2
Pb	0.34	0.11 to 1.1	0.08	less than 0.05 to 0.10
Zn	0.45	0.23 to 1.1	0.32	0.18 to 0.57

*Secondary settling pond with 0.5 hour retention time.

Filtration

Filtration is accomplished by the passage of water through a physically restrictive medium with the resulting deposition of suspended particulate matter. Typical filtration applications include polishing units and pretreatment of input streams to reverse osmosis and ion exchange units. Filtration is a versatile method in that it can be used to remove a wide range of suspended particle sizes.

Filtration processes can be placed in two general categories: (1) surface filtration devices, including microscreens and diatomaceous-earth filters and (2) granular media filtration, or in-depth filtration devices such as rapid sand filters, slow sand filters, and granular media filters.

Microscreens are mechanical filters which consist of a horizontally mounted rotating drum. The periphery of the drum is covered by fabric woven of stainless steel or polyester, with aperture sizes from 23 to 60 micrometers. Microscreens have found fairly widespread process application for concentrate dewatering, but are less used in wastewater treatment applications because of sensitivity to solids loadings and the relatively low filtration rates required to prevent chemical floc shearing and subsequent filter penetration.

Diatomaceous earth (DE) filters have been applied to the clarification of secondary sewage effluent at pilot scale and they produce a high quality effluent. However, they are relatively expensive and appear unable to handle the solids loadings encountered in this industry.

Next to gravity sedimentation, granular media filtration is the most widely used process for the separation of solids from wastewater. Most filter designs use a static bed with vertical flow, either downward or upward, using gravity or pressure as the driving force.

Slow sand filters are single, medium-gravity granular filters without a means of backwashing. The filter is left in service until the head loss reaches the point where the applied effluent rises to the top of the filter wall. Then the filter is drained and allowed to partially dry, and the surface layer of sludge is manually removed. Such filters require very large land areas and considerable maintenance. For these reasons, they are not competitive tertiary treatment processes other than for small package plants.

Rapid sand filters are much the same as slow sand filters in that they are composed of a single type of granular medium which is drained by gravity and hydrostatic pressure. The primary difference between the two is the provision for backwashing of the rapid sand filter by reversing the flow through the filter. During filter backwashing, the media (bed) is fluidized and settles with the finest particles at the top of the bed. As a result, most of the solids are removed at or near the surface of the bed. Only a small portion of the total voids in the bed are used to store particulates, and head loss increases rapidly. Despite this disadvantage, rapid sand filters are relatively common in potable water supply treatment plants.

Effective filter depth can be increased by the use of two or more types of granular media. Granular media filters typically use coal (specific gravity about 1.6), silica sand (specific gravity about 2.6), and garnet (specific gravity about 4.2) or ilmente (specific gravity about 4.5), with total media depths ranging from about 50 cm (20 inches) to about 125 cm (48 inches).

Pressure filters are often advantageous in waste treatment applications for the following reasons: (1) pressure filters can operate at higher heads than are practical with gravity filter designs, thus increasing run length and operational flexibility; (2) the ability to operate at higher head losses reduces the amount of wash water to be recycled; and (3) steel shell package units are more economical in small and medium-size plants (Reference 8).

Whenever possible, designs should be based on pilot filtration studies using the actual wastewater. Such studies are the only way to assure: (1) representative cost comparisons between different filter designs capable of equivalent performance (i.e., quantity filtered and filtrate quality); (2) selection of optimal operating parameters such as filter rate, terminal head loss, and run length; (3) effluent quality; and (4) determining effects of pretreatment variations. Ultimate clarification of filtered water will be a function of particle size, filter medium porosity, filtration rate, and other variables.

Granular media filtration has consistently removed 75 to 93 percent of the suspended solids from lime treated secondary sanitary effluents containing from 2 to 139 mg/l of suspended solids (Reference 9). One lead/zinc complex is currently operating a pilot-scale filtration unit to evaluate its effectiveness in removing suspended solids and nonsettleable colloidal metal hydroxide flocs from its combined mine/mill/smelter/ refinery wastewater. Preliminary data indicate that the single medium pressure filter operated at a hydraulic loading of 2.7 to 10.9 $l/sec/m^2$ (4 to 16 $gal/min/ft^2$) is capable of removing 50 to 95 percent of the suspended solids and 14 to 82 percent of the metals (copper, lead, and zinc) contained in the waste stream. Final suspended solids concentrations which have been attained are within the range of less than 1 to 15 mg/l. Optimum filter performance has been attained at the lower hydraulic loadings; performance at the higher hydraulic loadings appears to degrade significantly.

A full-scale granular media filtration unit is currently in operation at molybdenum Mine/Mill 6102. The filtration system consists of four individual filters, each composed of a mixture of anthracite, garnet, and pea gravel. This system functions as a polishing step following settling, ion exchange, lime precipitation, electrocoagulation, and alkaline chlorination. Since its startup in July 1978, the filtration unit has been operating at a flow of 63 liters/second (1,000 gallons/minute), and monitoring data from November 1979 to August 1980 have demonstrated significant reductions of TSS, Mo, Cu, Pb, Cd, and Zn. Suspended solids concentrations have been reduced from an average 34.7 mg/l to less than 11.3 mg/l. Zinc removals from 0.2 mg/l (influent) to 0.05 mg/l (effluent) and iron removals of 0.2 mg/l (influent) to 0.09 mg/l (effluent) have also been achieved.

A pilot-scale study of mine drainage treatment in Canada has demonstrated the effectiveness of filtration. Pre-filtration treatment consisted of lime precipitation, flocculation, and clarification. Polishing of the clarifier overflow by sand filtration further reduced the concentration of lead (extractable) from 0.25 mg/l to 0.12 mg/l, zinc from 0.37 mg/l to 0.19 mg/l, copper from 0.05 mg/l to 0.04 mg/l, and iron from 0.23 mg/l to 0.17 mg/l. For further discussion of this subject, refer to the discussion of Pilot- and Bench-Scale Treatment Studies, later in this section.

Also, slow sand filters are used on a full-scale basis at Mine/Mill 1131 to further polish tailing pond effluent prior to final discharge.

Recovery of metal values contained in suspended solids may, in some cases, offset the capital and operating expenses of filter systems. For example, filtration is used to treat uranium mill tailings for value recovery through countercurrent washing. In this instance, the final washed tail filter cake is reslurried for transport to the tailing pond.

Adsorption

Adsorption on solids, particularly activated carbon, has become a widely used operation for purification of water and wastewater. Adsorption involves the interphase accumulation or concentration of substances at a surface or interface. Adsorption from solution onto a solid occurs as the result of one of two characteristic properties for a given solvent/solute/solid system. One of these is the lyophobic (solvent-disliking) character of the solute relative to the particular solvent. For example, the more hydrophilic a substance is, the less likely it is to be adsorbed, and the reverse is true.

A second characteristic property of adsorption results from the specific affinity of the solute for the solid. This affinity may be either physical (resulting from van der Waal's forces) or chemical (resulting from electrostatic attraction or chemical interaction) in nature.

The best known and most widely employed adsorbent at present is activated carbon. The fact that activated carbon has an extremely large surface area per unit of weight (on the order of 1,000 square meters per gram) makes it an extremely efficient adsorptive material. The activation of carbon in its manufacture produces many pores within the particles, and it is the vast area of the walls within these pores that accounts for most of the total surface area of the carbon. In addition, due to the presence of carboxylic, carbonyl, and hydroxyl group residuals fixed on its surfaces, activated carbon also can exhibit limited ion exchange capabilities.

Granular activated carbon is generally preferred to the powdered form, due to dust and handling problems which accompany the latter. The commercial availability of a high activity, hard, dense, granular activated carbon made from coal, plus the development of multiple-hearth furnaces for on-site regeneration of this type of carbon, have drastically reduced the cost of granular activated carbon for wastewater treatment. Although powdered carbon is less expensive, it can only be used on a once-through basis and, subsequently, must be removed from the waste stream in some manner (e.g., filtration or settling).

A number of carbon-contacting system designs have been employed in other industries. Basic configurations include upflow or downflow, by gravity or pump pressure, with fixed or moving beds, and single (parallel) or multi-stage (series) unit arrangements. The most important design parameter is contact time. Therefore, the factors which are critical to optimum performance are flow rate and bed depth. These factors, in turn, must be determined from the rate of adsorption of impurities from the wastewater.

Activated carbon presently finds application in purification of drinking water and treatment of domestic, petroleum-refining, petrochemicals, and organic chemical wastewater streams. Compounds which are readily removed by activated carbon include aromatics, phenolics, chlorinated hydrocarbons, surfactants, organic dyes, organic acids, higher molecular weight alcohols, and amines. This technology also removes color, taste, and odor components in water. In addition, the potential of activated carbon to adsorb selected metals has been evaluated on both pure solutions and wastewater streams. The removal efficiencies range from slight to very high, depending on the individual metals, example of metals removal by activated carbon is presented in the tabulation that follows. This list is a summary of removal capabilities observed at three automobile wash establishments employing carbon adsorption for wastewater reclamation.

Metal	Concentration (mg/l-total)	
	Initial	Final
Cd	0.015 to 0.034	less than 0.005
Cr	0.01 to 0.125	less than 0.01
Cu	0.04 to 0.15	less than 0.01 to 0.02
Ni	0.045 to 0.16	less than 0.01 to 0.04
Pb	0.32 to 1.32	less than 0.02
Zn	0.382 to 1.49	0.02 to 0.417

In general, the literature indicates significant quantities of from wastewater by activated carbon. Removal of Cu, Cd, and Zn appears to be highly variable and dependent upon wastewater characteristics, while metals such as Ba, Se, Mo, Mn, and W are reported to be only poorly removed by activated carbon. The removal mechanism is thought to involve both adsorption and filtration within the carbon bed.

In addition to metals, other waste parameters of major interest in the ore mining and dressing industry are cyanide and phenolics. The use of granular carbonaceous material to catalyze the oxidation of cyanide to cyanate by molecular oxygen has been demonstrated (References 10, 11, and 12). The efficiency of cyanide destruction in this manner is reportedly improved when the cyanide is present as a copper cyanide complex (Reference 10). Application of this technology to treatment of copper-plating waste having an initial cyanide concentration of 0.315 to 4.0 mg/l has resulted in a final effluent concentration of 0.003 to 0.011 mg/l. Flow rate through the carbon bed was found to be 0.45 l/sec/m³ (0.2 gpm/ft³).

Phenolics have also been demonstrated to be readily removed by activated carbon in many industrial applications. However, little information is available relative to removal of phenolics at concentrations characteristic of milling wastewater (i.e., less than 3 mg/l).

Cyanide Treatment

Depressing agents are commonly used in the flotation of metal ores to assist in the separation of minerals with similar floatabilities. As discussed previously, cyanide, either as calcium cyanide flake or as sodium cyanide solution, is widely used as a depressant for iron sulfides, arsenopyrite, and sphalerite during flotation of base metals, and ferroalloys. Cyanide is also used in processing lode gold and silver ores by the cyanidation process, a leaching process.

The use of cyanide in these milling processes results in its presence in mill tailings and wastewater. The maximum theoretical concentration of total cyanide in untreated mill wastewater, based on reported reagent consumption and water use, is approximately 1.3 mg/l for flotation operations and 114 mg/l for gold cyanidation operations. In practice, however, cyanide levels below the theoretical maximum are observed. (Refer to Section VI, Wastewater Characteristics.)

An additional source of cyanide-bearing wastewater is underground mines which backfill stopes with the sand fraction of mill tailings. Residual cyanide is found in tailings from flotation circuits using sodium cyanide as a depressant. At least two lead/ zinc facilities cyclone these tailings to separate the heavy sand fraction from slimes and then sluice the sands to backfill mined-out stopes. Overflow from the backfilled stopes introduces cyanide to the mine drainage.

The dissociation of simple cyanide salts in water and the subsequent hydrolysis of the cyanide ion leads to the formation of hydrocyanic acid (HCN). The relative amounts of free cyanide ion to HCN are dependent on pH. For example, at pH 7, the ratio of cyanide (CN^-) to HCN is 0.005 to 1; at pH 11, the ratio of CN^- to HCN is 50 to 1.

In addition to the presence of free cyanide ion and hydrocyanic acid, it has been suggested that the predominant cyanide species found in flotation mill wastewater are metal-cyanide complexes. Willis and Woodcock (References 13 and 14) have demonstrated the presence of copper-cyanide complexes in flotation circuits, with cupro-cyanides ($Cu(DN)_3^{-2}$ and/or CuCN being the predominant complexes formed.

Although only the presence of copper-cyanide complexes in flotation circuits has been shown, the presence of other transition metals in the float circuit may present situations favorable to the formation of additional metal-cyanide complexes. These additional complexes include zinc cyanides ($Zn(CN)_4-2$ and/or $Zn(CN_2)$, and iron-cyanides ($Fe(CN)_6^{-2}$ and/or $Fe(CN)_6^{-3}$). Indirect evidence for their existence has been presented by two domestic mills. Both operations have inferred the existence of iron cyanide complexes in mill tailings based on the presence of residual cyanide in the effluents from laboratory and pilot-plant treatability studies (Reference 15 and 16).

Three options available to eliminate cyanide from mill effluents are: (1) in-process control, (2) use of alternative depressants, and (3) treatment. The particular option or combination depends on process type, existing controls, the availability and applicability of alternatives, plant economics, and personal preference of the plant operator. In-process control and alternative reagent use have been discussed; treatment is discussed here.

Sophisticated technology for the destruction of cyanide is not employed at most domestic mine/mill operations which use cyanide. Such technology is generally not necessary because in-process controls and retention of mill tailings in tailing ponds have reduced cyanide concentrations to less than detectable levels in the final effluents. The mechanism of cyanide decomposition within a tailing pond is thought to involve photo-decomposition by ultraviolet light (Reference 17) and biochemical oxidation. For this reason, elevated levels of cyanide in the final effluent (tailing-pond decant) are some times observed during winter months, when daylight hours are at a minimum and ice sometimes covers the tailing pond.

Because of increasingly stringent regulation of cyanide in industrial wastewater discharges during recent years, a number of domestic and foreign mine/mill operations have investigated and implemented sophisticated technology for cyanide destruction. Treatment technologies which have been investigated and/or employed by various industries for the destruction of cyanide are listed below:

1. Chemical oxidation
    - Alkaline chlorination (calcium, sodium, or magnesium hypochlorite)
    - Gaseous chlorine
    - Permanganate
    - Ozone
    - Hydrogen or sodium peroxide
2. Electrolysis
3. Biological Degradation
4. Carbon-Bed Oxidation
5. Destruction by Gamma Irradiation
6. Physical Treatment
    - Ion exchange
    - Reverse osmosis
7. Ferrocyanide Precipitation

Of the technologies listed above, alkaline chlorination, hydrogen peroxide, and ozonation appear to be best suited for use in the ore mining and dressing industry. They most readily lend themselves to the treatment of high volume, relatively low concentration waste streams at reasonable cost. Free cyanide and cadmium, copper, and zinc-cyanide complexes can be destroyed by these treatment technologies. However, it is uncertain in the ore industry whether cyanide complexes (such as nickel cyanide and iron cyanide) are attacked or destroyed by chlorine or ozone. Thus, the effectiveness of these technologies is dependent on the specific nature of the wastewater treated.

Alkaline Chlorination Theory. The kinetics and mechanisms of cyanide destruction have been described in the literature (References 18 through 23). Destruction is accomplished by oxidation of free cyanide (CN^-) to cyanate (CNO^-) and, ultimately, to CO_2 and N_2. Destruction of metal-cyanide complexes (e.g., CuCN) is accomplished by oxidation of the complex anion to form the metal cation and free cyanide. The probable reactions in the presence of excess chlorine are:

$$Cl_2 + CN^- + 2NaOH \text{ ---> } CNO^- + H_2O + 2NaCl$$

and

$$3Cl_2 + 2CuCN + 8NaOH \text{ ---> } 2NaCNO + 2Cu(OH)_2 + 6NaCl + 2H_2O$$

Rapid chlorination at a pH above 10 and a minimum of 15-minutes contact time are required to oxidize 0.45 kilogram (1 pound) of cyanide to cyanate with 2.72 kilograms (6 pounds) each of sodium hydroxide (caustic soda) and chlorine. If metal-cyanide complexes are present, longer detention periods may be necessary.

An alternative chlorination technique involves the use of sodium hypochlorite (NaOCl) as the oxidant. Reactions with sodium hypochlorite are similar to those of chlorine except that there is no caustic requirement for destruction of free cyanide in the oxidation stages. However, alkali is required to precipitate metal- cyanide complexes as hydroxides. Reactions of the free cyanide and the metal-cyanide complex with hypochlorite are:

$$NaOCl + CN^- \text{ ---- } CND^- + NaCl$$

and

$$3NaOCl + 2CuCN + 2NaOH + H_2O \text{ --- } 2NaCNO + 2Cu(OH)_2 + 3NaCl$$

To oxidize cyanide to cyanate, a 15 percent solution of sodium hypochlorite is required at a dosage rate ranging from 2.72 kilograms (6 pounds) to 13.5 kilograms (30 pounds) of sodium hypochlorite per 0.45 kilogram (1 pound) of cyanide is required to oxidize cyanide.

Complex destruction of cyanate requires a second oxidation stage with an approximate 45-minute retention time at a pH below 8.5. The theoretical reagent requirements for this second stage are 1.84 kilograms (4.09 pounds) of chlorine and 0.51 kilogram (1.125 pounds) of caustic per 0.45 kilogram (1 pound) of cyanide. Actual reagent consumption and choice of reagent will be dependent on process efficiency, residual chlorine levels from the first oxidation stage, optimization through pilot-scale testing, temperature, etc. The overall reaction for the second stage is:

$$3Cl_2 + 2CNO^- + 6\ NaOH \text{ ----> } 2HCO_3^- + N_2 + 6NaCL + 2H_2O$$

Note that the intermediate reaction product, carbon dioxide, reacts with alkalinity in the water to form bicarbonate.

Advantages to the use of alkaline chlorination include relatively low reagent costs, applicability of automatic process control, and experience in its use in other industries (e.g., electroplating). Major disadvantages are the potential health and pollution hazards associated with its use, such as worker exposure to chlorine gas (if gas is used) and cyarogen chloride (byproduct gas), the potential for production of harmful chloramines and chlorinated hydrocarbons, and the presence of high chlorine residual levels in the treated effluent.

Ozonation Theory. Because of the disadvantages associated with alkaline chlorination, ozonation is receiving a great deal of attention as a substitute technique for cyanide destruction. Oxidation of cyanide to cyanate with ozone requires approximately 0.9 kilogram (2 pounds) of ozone per 0.45 kilogram (1 pound) of cyanide, and complete oxidation requires 2.25 kilograms (5 pounds) of ozone per 0.45 kilogram (1 pound) of cyanide. Cyanide oxidation to cyanate is very rapid (10 to 15 minutes) at pH 9 to 12 and practically instantaneous in the presence of trace amounts of copper. Thus, the destruction of cyanide to cyanate in mill wastewater containing copper cyanide complexes can be expected to proceed rapidly.

The reaction mechanism for the destruction of cyanide to cyanate is generally expressed as:

$$CN^- + O_3 \text{ ----> } CNO^- + O_2$$

The reaction mechanism for the subsequent reaction, destruction of cyanate, has not been positively identified. Proposed mechanisms include (Reference 24 and 25):

$$2CNO^- + O_3 + H_2O \text{ ---> } 2HCO_3 + N_2 + 3O_2$$
$$CNO^- + OH^- + H_2O \text{ ----> } Co_3{}^{-2} + NH_3$$

or

$$CNO^- + NH_3 \text{ ----> } NH_2\text{-}\ CO\text{-}\ NH_2$$

Regardless of the actual mechanism, destruction of cyanate can be accomplished in approximately 30 minutes (Reference 26).

Hydrogen Peroxide Theory. Two processes for the oxidation of cyanide with hydrogen peroxide (H_2O_2) have been investigated on a limited scale. The first process involves the reaction of hydrogen peroxide with cyanide at alkaline pH in the presence of a copper catalyst. The following reactions are observed:

$$CN^- + H_2O_2 \text{ ----> } CNO^- + H_2O$$
$$CNO^- + 2H_2O_2 \text{ ----> } NH_4 + CO_3{}^{-2}$$

The second process, known as the Kastone process, uses a formulation containing 41 percent hydrogen peroxide, trace amounts of catalyst and stabilizers and formaldehyde. The cyanide wastes are heated to 129 C (248 F), treated with three to four parts of oxidizing solution and two to three parts of a 37 percent solution of formaldehyde per part of sodium cyanide, and agitated for one hour. Principal products from the reaction are cyanates, ammonia, and glycolic acid amide. Complete destruction of cyanates requires acid hydrolysis (Reference 23).

Cyanide Treatment Practices. As discussed, treatment specifically designed for cyanide treatment has not been widely installed in this industry. However, investigations of treatment techniques specific to cyanide reduction have been conducted.

Extensive laboratory and pilot-plant tests on cyanide destruction in mill wastewater were conducted by molybdenum Mill 6102 for the development of a full scale waste treatment system, which was brought on-line during July 1978. Testing was aimed at identifying treatment to achieve a 1 July 1977 permit limitation of 0.025 mg/l.

Ozonation tests in the laboratory showed substantial destruction of cyanide, as the data in Table VIII-2 show. The target level of less than 0.025 mg/l of cyanide was not achieved, however, and tests of ozonation under pilot-plant conditions showed even less favorable results. Ozonation did, however, result in substantial removals of cyanide and manganese.

Laboratory chlorination tests (also at Mill 6102) indicated that removal of cyanide to 0.025 mg/l or less could be achieved under the proper conditions. As the data in Table VIII-3 show, chlorine doses in excess of stoichiometric amounts were required, and pH was found to be a major determinant of treatment effectiveness. Results on a pilot-plant scale were less favorable, but improved performance in the full-scale treatment system is anticipated through use of a retention basin in which additional oxidation of cyanide by residual chlorine can take place.

Mill 6102 has built a treatment system employing lime precipitation, electrocoagulation-flotation, ion exchange, alkaline chlorination, and mixed media filtration. This is followed by final pH adjustment. The alkaline chlorination system includes on-site generation of sodium hypochlorite by electrolysis of sodium chloride. The hypochlorite is injected into the wastewater following the electrocoagulation-flotation process and immediately preceeding the filtration unit. At this point in the system, some cyanide removal has been realized incidental to the lime precipitation-electrocoagulation treatment. The first four months of operating data show the concentration of cyanide at 0.09 mg/l prior to the electrocoagulation unit. Concentrations of cyanide progressively decreased from a 0.04 mg/l (electrocoagulation effluent) to less than or equal to 0.01 mg/l after filtration, and less than 0.01 mg/l after the final retention pond. Mill personnel expect this removal efficiency to continue throughout the optimization period of the system. The problem of chlorine residuals at elevated levels has not been resolved.

Control and treatment of cyanide at a Canadian lead/zinc operation, Mill 3144, is achieved by segregation of the cyanide bearing waste streams and subsequent destruction of the cyanide by alkaline chlorination. Waste segregation is practiced to reduce the volume and solids loading of the cyanide-bearing waste streams. The waste stream treated in the alkaline chlorination treatment plant comprises about 30 percent of the total tailings volume ultimately discharged, or approximately 1,000 cubic meters (300,000 gallons) of tailings per day. This waste stream has an initial cyanide concentration of approximately 60 to 70 mg/l.

The initial design of the alkaline chlorination treatment plant was based on extensive laboratory and pilot-plant studies. Final design modifications, made in 1975, were based on a requirement to comply with an amended discharge permit limitation of 0.5 mg/l cyanide (total). Because some cyanide is present in the 3,000 cubic meters (700,000 gallons) per day of tailings not treated, the alkaline chlorination treatment plant is operated with the goal of destroying the cyanide contained in that portion of wastewater being treated. The composite waste stream should then meet the permit limitation.

The treatment plant includes three FRP (fiber-reinforced-plastic) tanks, measuring 3 by 3 meters (10 by 10 feet), which operate in series, as reaction chambers. Chlorine is added at a rate of 540 to 680 kilograms (1,200 to 1,500 pounds) per day from a chlorinator having a capacity of 900 kilograms (2,000 pounds) per day of chlorine. The pH of the combined waste stream is maintained between 11 and 12 by lime addition to the incoming waste stream. Process controls include pH and ORP (oxidation/reduction-potential) recorders and a magnetic flow meter.

The average cyanide concentration of the total tailing effluent discharge between July and December 1975 was 0.18 mg/l of cyanide (total). This compares to the average concentration of 4.72 mg/l of cyanide (total) in the discharge prior to installation of the treatment plant. Performance data for the alkaline-chlorination treatment plant at Mill 3144 are presented in the following table (industry data):

Source	Total Cyanide (mg/l)*
Chlorination-Plant Feed	68.3
Chlorination-Plant Discharge	0.13
Mine Drainage (Overflow from Backfill)	0.06
Total Combined Tailings	0.07

*Average of daily samples taken during September 1975.

Government regulations in the USSR presently limit the discharge of cyanide waste from ore milling operations to 0.1 mg/l of cyanide. A more stringent limitation of 0.05 mg/l applies when cyanide-bearing wastewater is discharged to surface waters inhabited by fish (Reference 27).

Two references in the literature described the use of alkaline chlorination in the Soviet Union for treatment of cyanide in ore-milling wastewater (References 28 and 29). At one mill, the effluent, containing 95 mg/l of cyanide ion, was treated with chlorine at a dosage rate of approximately 10 parts chlorine to 1 part cyanide. It was claimed that the cyanide was completely destroyed. A second mill treated the overflow from copper and lead thickeners with calcium hypochlorite. The overflow contained more than 45 mg/l of cyanide ion. The cyanide concentration of the treated effluent was reported to be less than 1 mg/l, although difficulties were experienced in the oxidation stage. Similar difficulties with the use of calcium hypochlorite have been reported elsewhere (Reference 17).

Research conducted by the Air Force Weapons Laboratory on the destruction of iron cyanide complexes has resulted in development of a pilot-scale process to treat electroplating and photographic processing wastes. Briefly, the process employs ozonation at elevated temperatures and ultraviolet irradiation to reduce cyanide concentrations in the effluent to below detectable levels (Reference 30).

Reduction of cyanide in tailing pond decant water using hydrogen peroxide has been practiced on an experimental basis at Mill 6101. Although earlier monitoring data had shown cyanide to be reliably absent from the effluent (less than 0.02 mg/l), recent data using EPA approved analytical procedures indicated that, during the colder months, elevated levels of cyanide (up to approximately 0.09 mg/l) may occur. To reduce these levels, mill 6101 has experimented with a very simple peroxide treatment system with modest success.

Treatment is provided by dripping a hydrogen peroxide solution from a drum into the channel carrying wastewater from the tailing pond to the secondary settling pond. Mixing is by natural turbulence in the channel, and the peroxide addition rate is manually adjusted periodically based on the effluent cyanide concentration. Results to date indicate that this simple treatment system achieves cyanide removals on the order of 40 percent.

Treatment of Phenols

Several phenolic compounds are used as reagents in floation mills. These compounds, which include Reco, AEROFLOAT 31 and 242, AERO Depressant 633, cresylic acid, and diphenyl guanidine, find specific application as promoters and frothers in the flotation process.

Phenol-based compounds can be a significant source of phenolics in mill process wastewater. The degree of control exerted over reagent addition, uniformity in ore grade, and mill operator preferences could affect residual phenol levels in the flotation circuit. The surface-active nature of frothers and promoters, coupled with the volatility of many phenolic compounds, further complicates the theoretical prediction of phenol concentrations in flotation-mill wastewater streams.

A more detailed account of reagent use, including phenolic-based compounds and their presence in mill process wastewater, is provided in Section VI, Wastewater Characteristics.

Several methods are available for treating phenolic wastes, including:

1. Chemical Oxidation
    - Chlorine dioxide
    - Hydrogen peroxide
    - Ozone
    - Potassium permanganate
2. Biological Oxidation
3. Carbon Adsorption
4. Aeration
5. Ultraviolet Irradiation
6. Incineration
7. Recovery

The specific technology applied depends on the chemical characteristics of the waste stream, the discharge concentrations required, and the economics of implementation. However, the low-concentration, high-volume phenolic wastes generated in this industry are best treated by chemical oxidation or aeration.

Chemical Oxidation Theory. Chemical oxidizing agents react with the aromatic ring of phenol and phenolic derivatives, resulting in its cleavage. This cleavage produces a new organic compound (a straight-chain compound), which still exerts a chemical oxygen demand (COD). Complete destruction of the organic compound (conversion to CO_2 and H_2O) and reduction in COD requires either additional chemical oxidation or other treatment (e.g., biological oxidation).

Complex wastewater may require an additional oxidizing agent. As an example, hydrogen peroxide will react with sulfides, mercaptans, and amines in addition to phenolic compounds. The total consumption of oxidizing agent is dependent on the type and concentration of oxidizable species present in the waste, the reaction kinetics, and the end products desired (i.e., straight-chain organic compounds or carbon dioxide and water). Therefore, it is difficult to predict actual reagent consumption without treatability studies. Some general guidelines may be given, however, for the various oxidizing agents available.

Chlorine Dioxide. Chlorine gas is considered unacceptable as an oxidizing agent because of the potential for forming toxic chlorinated phenols, as well as the potential safety hazards involved in handling the gas. As an alternative, chlorine dioxide (ClO_2) can be generated on-site from chlorine gas or hypochlorite and used as a relatively safe oxidizing agent. Chlorine dioxide reacts with phenol to form benzoquinone ($C_6H_4O_2$) within the pH range of 7 to 8 and at a reagent dosage of 1.5 parts of ClO_2 per part of phenol. At a pH above 10 and a dosage of 3.3 parts of ClO_2 per part of phenol, maleic acid and oxalic acid are formed, rather than benzoquinone.

Hydrogen Peroxide. In the presence of a metal catalyst (e.g., Fe^{++}, Fe^{+++}, Al^{+++}, Cu^{++}, and Cr^{++}), hydrogen peroxide (H_2O_2) effectively oxidizes phenols over a wide range of temperatures and concentrations. Investigations into the use of hydrogen peroxide show phenol removal efficiencies of 98+ percent at a dosage rate of one to two parts of H_2O_2 per part phenol and at an optimum pH between 3 and 5. Wastewater containing substituted phenols, such as cresylic acid, can increase the required peroxide dosage to four parts of H_2O_2 per part of substituted phenol. An approximate five-minute retention time is required to partially oxidize simple phenols. Either batch or continuous operation may be employed, with batch treatment preferred at flows less than 190 to 380 cubic meters (50,000 to 100,000 gallons) per day (Reference 31).

Ozone. Ozone, a very strong oxidizing agent, attacks a variety of materials, including phenols. Because of its poor selectivity, ozonation is generally used as a polishing step after conventional treatment processes which remove gross suspended solids and nontoxic organic compounds.

Ozone will completely oxidize phenols to carbon dioxide and water if a sufficient retention time and enough ozone are provided. In practice, however, the reaction is allowed to proceed only to intermediate, straight-chain organic compounds. Ozone requirements for the partial destruction of phenols range from one to five parts per part of phenol. The actual ozone demand is a function of phenol concentration, pH, and retention time. For example, reduction of phenol in a particular wastewater from 2,500 mg/l to 25 mg/l, at a pH of 11, has been found to require 1.7 parts of ozone per part of phenol and a 60-minute retention time. The same waste, when treated at a pH of 8.1, required 5.3 parts of ozone per part of phenol and a 200-minute retention time to achieve similar reductions in phenol (Reference 63). The efficiency of ozonation appears to increase with decreasing phenol concentration. Operating data from a full-scale ozonation system treating 1,500 cubic meters (400,000 gallons) per day of wastewater from a Canadian refinery show ozone requirements of one part per part of phenol to reduce phenol reductions to 0.003 mg/l at dosage rates ranging from 1.5 to 2.5 parts of ozone per part of phenol (References 31 and 32).

Potassium Permanganate. Paint-stripping and foundry wastes containing 60 to 100 mg/l of phenol have been treated with potassium permanganate ($KMnO_4$). Phenol reductions to less than 1 mg/l are reported. The destruction of simple phenol by potassium permanganate is expressed as:

$$3C_6H_6O + 28KMnO_4 + 5H_2O \text{ --- } 18CO_2 + 28KOH + 28MnO_2$$

Based on this expression, the theoretical dosage for complete destruction of phenol is 15.7 parts of $KMnO_4$ per part of phenol. However, dosages of only six to seven parts of $KMnO_4$ per part of phenol have been reported to be effective in destroying the aromatic-ring structure. Optimum pH for permanganate destruction of phenol is between seven and ten (References 31 and 33).

A disadvantage to the use of potassium permanganate is the generation of a manganese dioxide precipitate, which settles as a hydrous sludge. Clarification and sludge disposal may be required, resulting in additional equipment and maintenance costs.

Aeration Theory. Limited phenol removal may be obtained in ponds or lagoons by simple aeration. In general, forced aeration is more effective in reducing phenol levels than is passive aeration. Field studies have shown that, at an initial concentration of 15 mg/l of phenol, wastewater phenol levels can be reduced to approximately 1 mg/l after 30 hours of forced aeration and after 70 hours of passive aeration (Reference 31). The mechanisms for removal of phenols in ponds is not well understood, but probably includes degradation by biological action and ultraviolet light, and simple air stripping.

Phenol Treatment Practices. The only treatment for phenols used in the ore mining and dressing industry is aeration. In practice, phenol reduction is incidental to treatment of more traditional design parameters (i.e, heavy metals, suspended solids, etc.). At Mill 2120, phenol concentrations in the tailing-pond influent and effluent average 0.031 mg/l and 0.021 mg/l, respectively. Similar results are noted at Mill 2122, where phenol concentrations in the tailing-pond influent and effluent average 0.26 mg/l and 0.25 mg/l, respectively. Data from samples collected at Mill 2117 show phenol reductions from 5.1 mg/l of phenol in the raw tailings to 0.25 mg/l of phenol in the tailing-pond overflow.

Sulfide Precipitation of Metals

The use of sulfide ions as a precipitant for removal of heavy metals can accomplish more complete removal than hydroxide precipitation. Sulfide precipitation is widely used in wastewater treatment in the inorganic chemicals industry for the removal of heavy metals, especially mercury. Effective removal of cadmium, copper, cobalt, iron, mercury, manganese, nickel, lead, zinc, and other metals from mine and mill wastes show promise by treatment with either sodium sulfide or hydrogen sulfide. The use of this method depends somewhat on the the availability of methods for effectively removing precipitated solids from the waste stream, and on removal of the solids to an environment where reoxidation is unlikely.

Several steps enter into the process of sulfide precipitation:

1. Preparation of sodium sulfide. Although this product is often in abundence as a byproduct it can also be made by the reduction of sodium sulfate, a waste product of acid-leach milling. The process involves an energy loss in the partial oxidation of carbon (such as that contained in coal) as follows:

$$Na_2SO_4 + 4C \text{ --- } Na_2S + 4CO \text{ (gas)}$$

2. Precipitation of the pollutant metal (M) in the waste stream by an excess of sodium sulfide:

$$Na_2S + MSO_4 \text{ -- } MS \text{ (precipitate)} + Na_2SO_4$$

3. Physical separation of the metal sulfide in thickeners or clarifiers, with reducing conditions maintained by excess sulfide ion.

4. Oxidation of excess sulfide by aeration:

$$Na_2S + 2O_2 \text{ --- } Na_2SO_4$$

This process usually involves iron as an intermediary and, as illustrated, regenerates unused sodium sulfide to sodium sulfate.

In practice, sulfide precipitation can be best applied when the pH is sufficiently high (greater than about eight) to assure generation of sulfide, rather than bisulfide ion or hydrogen sulfide gas. It is then possible to add just enough sulfide, in the form of sodium sulfide, to precipitate the heavy metals present as cations. Alternatively, the process can be continued until dissolved oxygen in the effluent is reduced to sulfate and anaerobic conditions are obtained. Under these conditions, some reduction and precipitation of molybdates, uranates, chromates, and vanadates may occur; however, ion exchange may be more appropriate for the removal of these anions.

Because of the toxicity of both the sulfide ion and hydrogen sulfide gas, the use of sulfide precipitation may require both pre- and post-treatment and close control of reagent additions. Pretreatment involves raising the pH of the waste stream to minimize evolution of H_2S, which could cause odors and pose a safety hazard to personnel. This may be accomplished at essentially the same point as the sulfide treatment, or by addition of a solution containing both sodium sulfide and a strong base (such as caustic soda). The sulfides of many heavy metals, such as copper and mercury, are sufficiently insoluble to allow essentially complete removal with low residual sulfide levels. Treatment for these metals with close control on sulfide concentrations could be accomplished without the need for additional treatment. Adequate aeration should be provided to yield an effluent saturated with oxygen.

Sulfide precipitaition is presently practiced at most mercury-cell chloralkali plants to control mercury discharges. In this application, treatment with sodium sulfide is commonly followed by filtration and typically results in the reduction of mercury concentrations from 5 to 10 mg/l to 0.01 to 0.05 mg/l. Although lead is also present in the waste streams treated with sulfide, its concentration is not often measured. The limited available data indicate that lead is also removed effectively by sulfide precipitation (Reference 34).

Sulfide is also used in the treatment of chemical-industry waste streams bearing high levels of chromates. It is reported that, after sulfide treatment and sedimentation, levels of hexavalent chromium consistently below 1 microgram per liter and total chromium between 0.5 and 5 micrograms per liter are achieved in the effluent (Reference 35). Other sources report that sulfide precipitation can achieve effluent levels of 0.05 mg/l arsenic, 0.008 mg/l cadmium, 0.05 mg/l selenium, and "complete" removal of zinc (References 35, 36, 37, and 38).

Sulfide precipitation is not employed in the domestic metal-ore mining and milling industry at present. However, the use of sulfide for removal of copper, zinc, and manganese from acid-mine drainage has been evaluated both theoretically and experimentally (References 35 and 39). A field study of mine drainage treatment in Colorado demonstrated that greater than 99.8 percent removal of metals was attained by treatment which consisted of partial neutralization with lime, followed by sulfide addition and settling. The treated effluent attained in this manner contained 0.2 mg/l zinc, 0.4 mg/l manganese, and less than detectable concentrations of copper and arsenic. However, it was also noted that the standard neutralization with lime and settling produced similar results.

Asbestos Treatment

The term "asbestos" has many definitions (Reference 40). The EPA's Effluent Guidelines Division chose to define asbestos as chrysotile for the purpose of this program. (For the rationale for this choice refer to page nine of Reference 40.)

The main source of asbestos fibers in this industry is the milling and beneficiation of copper, iron, nickel, molybdenum and zinc ores. The total-fiber counts made from samples collected at

Data on asbestos fiber removal in this industry is very limited. However, the physical treatment processes used and consistent fiber morphology make data from municipal and other industries applicable. Two good data sources are the Duluth, Minnesota potable water treatment plant and the chlorine/caustic (chloralkalai) industry.

Duluth Treatment Plant. Extensive pilot-scale studies on removal of asbestiform minerals from Lake Superior water were conducted by Black and Veatch Consulting Engineers under joint sponsorship of the EPA and the Army Corps of Engineers (References 41, 42, and 43). Filtration processes investigated included filtration, pressure diatomaceous earth (DE) filtration, and vacuum DE filtration.

Forms of asbestiform minerals evaluated in these studies include amphibole and chrysotile. A basic difference between these forms which influences treatment is that the treatment for the amphibole form usually carries important conclusions and recommendations from the pilot-scale studies on granular filtration, including:

1. Granular filtration is successful in removal of asbestiform fibers.

2. Sedimentation prior to filtration increases filter run length (i.e., time between backwashes) but does not increase fiber removal. (Note that untreated water for these studies was Lake Superior water, clean water relative to mine/mill wastewater. Sedimentation does remove some asbestos fiber in this industry and would be necessary to prevent filter clogging and frequent filter backwashing.)

3. Two-stage flash-mixing followed by flocculation is recommended for conditioning raw water prior to filtration.

4. Alum is a more effective coagulant than ferric chloride for Duluth raw water.

5. Nonionic polyelectrolyte is most effective in preventing turbidity breakthrough.

6. A positive-lead mixed media filter designed to operate

7. Backwash water should be discharged to a sludge lagoon, and supernatant should be returned to the treatment plant.

8. For large capacity plants, granular filtration is recommended over DE filtration. For small plants, DE filtration should also be considered.

Two kinds of DE filtration processes were studied in the pilot-scale studies, pressure filtration and vacuum filtration. Flow through both filtration systems ranged from 0.0006 to 0.001 m^3/sec (10 to 20 gal min). Both kinds of DE filters were operated in various ways to evaluate conditioning of DE with alum, cationic polymers, and anionic polymers. Single-step and twostep precoat were studied. Conditioned DE was used in filter precoat, as well as for body feed. Various grades of DE, from fine to coarse, were evaluated. Details of pilot-plant DE testing are reported in References 41, 42, and 43.

Vacuum DE filtration was found unsuitable for treating the raw Lake Superior water being tested because of the formation of air bubbles in the filter during filtration of cold water. (This is not to say that vacuum filtration would not be effective on warmer waters.) For the conditions experienced during the pilot-scale studies on potable Lake Superior water, it was concluded that pressure DE filtration is effective (i.e., reduces fiber content to 4 x 10^4 fibers/liter or less) with the addition of chemical aids to the precoat, the body feed, and the raw water.

Conditioning steps found to effect high removal of fibers consisted of the following:

1. Alum coatings or plain precoat with cationic polymer added to raw water
2. Anionic polymer added to precoat and alum-coated body feed
3. Filter precoat with medium-grade DE
4. Conditioning of DE with alum or soda ash, or with anionic polymer
5. Fine grade of DE for body feed
6. DE filter flow rate of approximately 41 liters/min/m^2 (1 gal/min/ft^2).

Pilot plant studies have been conducted on asbestos removal using synthetic asbestos suspensions containing approximately the same concentrations and size distributions as typical asbestos mine effluents (i.e., about 10^{12} fibers/liter) (Reference 44). In addition, pilot plant tests have been conducted on samples of asbestos-laden water from three locations in Canada. Treatment processes studied include plain sedimentation, sand filtration, mixed media (sand and anthracite) filtration, and DE filtration.

Effectiveness of the various treatment methods at pilot plants using synthetic asbestiform (chrysotile) particle-laden water is summarized in Table VIII-4 (Reference 44).

Data on pilot-plant asbestos removal from wastewater at specific locations are summarized in Tables VIII-5 and VIII-6. The authors conclude that sedimentation followed by mixed-media filtration is very effective for removing most of the asbestos from mine and processing plant effluents. Diatomite filters are even more effective, but may be more than what is necessary except where wastewater streams are discharged into water used as a drinking water source (Reference 44).

Based on the pilot-scale studies discussed previously (Reference 42), a 114,000-m^3/day (30,000,000-gal/day) granular filtration plant was designed and constructed for treating the water supply for the city of Duluth, Minnesota. This plant became operational in November 1976 (Reference 45). Figure VIII-1 presents a schematic diagram of the principal portion of this full-scale plant. The major components of the system are the mixed-media filtration beds containing anthracite, sand, and ilmenite. Each of the four filters has a capacity of 28,400 m^3/day (7,500,000 gal/day) at a design loading rate of 204 liters/min/m^2 (5.0 gal/min/ft^2).

Prior to filtration, raw Lake Superior water is flocculated and settled to increase fiber removal efficiency as well as to provide suspended solids separation. Coagulation facilities are three rapid-mix chambers. Anionic polymer is added in the first chamber, alum and caustic soda in the second chamber, and nonionic polymer in the third chamber. Flocculation and sedimentation are carried out in adjacent tanks. Effluent from the sedimentation basin flows directly to the mixed media filters described previously.

Filters are backwashed when head loss becomes greater than approximately 2.4 m (8 feet) or when effluent turbidity is greater than 0.2 JTU (Jackson Turbidity Units). Filter backwash water is sent to a storage tank and then to a settling lagoon, where alum and polymer are added to increase solids settling. Supernatant from the settling basin is sent back through the treatment system. Settled sludge from the settling basin is mechanically transferred to a sludge lagoon for further settling. Sludge is periodically removed from the lagoons and disposed of in a sanitary landfill. Decant water from the sludge lagoons is also returned to the treatment influent. The frequent freeze/thaw cycles experienced in northern Minnesota enhance water separation from the asbestos laden sludge. This phenomenon may not occur in other regions.

The Duluth plant is being monitored closely. Unpublished data fibers/liter by the full-scale mixed media filtration plant.

Chlorine/Caustic Industry. Treatment for the removal of asbestos from wastewater is practiced at a significant and growing number of facilities which produce chlorine and caustic soda by electrolysis in diaphragm cells. Treatment practices used include both sedimentation and filtration, with flocculants frequently used to enhance the efficiency of either process. At the chlorine/caustic facilities, asbestos removal is practiced on segregated, relatively low-volume waste streams which generally have high levels of suspended solids, consisting mostly of asbestos. Due to uncertainties in the analytical procedure, asbestos concentrations are not generally reported explicitly and must be inferred from TSS data.

At one chlorine/caustic facility in Michigan (Reference 34), a pressure leaf filter is used, together with flocculants, to treat a 1.5 liter/second (25 gal/min) stream to remove (by filtration) approximately 102 kg (225 lb) of asbestos per day. Effluent performance of this system, reported by the facility as "no detectable asbestos discharge," was verified by sample analysis, which shows a reduction of asbestos (total fiber) content from a concentration of greater than 5 x 10^9 fibers/liter in the filter influent to less than detectable (approximately 3 x 10^5 fibers/liter) in the filtered discharge.

Another facility removes asbestos from an intermittent flow totaling about 37.8 m^3 (10,000 gal)/day by sedimentation in a concerete sump with a volume of 327 m^3 (11,550 ft^3) and compartments to provide separate surge and settling chambers. With the addition of flocculants, this system reduces TSS (of which a significant fraction is asbestos) from about 3,000 mg/l to 30 mg/l.

Other facilities report the use of sedimentation technology (with varying degrees of efficiency), or the elimination of asbestos discharges by wastewater segregation and impoundment. Plans have been announced for additional use of filtration within the chlorine/caustic industry.

Review and comparison of pilot-scale results from treatment of raw water with granular filtration and DE filtration discussed earlier (Reference 42) indicate that similar results can be obtained from the two systems. Data in Tables VIII-5 through VIII-7 (from Reference 44), however, indicates that DE filtration may be more effective. An economic analysis of both types of systems has been conducted (Reference 42).

Practices in This Industry. No treatment systems in use in this industry are operated specifically for asbestos removal. However, asbestos is a suspended solid and, as discussed in Section VI, correlates very well with TSS in wastewater generated in this industry. Therefore, asbestos data taken at facilities designed and operated for TSS removal is indicative of the industries' current asbestos removal practices.

The sampling program was not designed to establish treatment efficiencies and samples are grabs and 24 hour-composites taken over short terms. However, the data obtained generally demonstrates the effectiveness of asbestos removal by existing facilities.

Table VIII-8 is a comparison of the total-fiber and chrysotile asbestos contents of the influent and effluent streams associated anions, such as Cl^- and SO_4^{-2}. Anions adsorb along with treatment systems at the facilities surveyed. The data indicate better removal of asbestos in mill water than in mine water.

For mill treatment systems consisting primarily of tailing ponds and settling or polishing ponds, some facilities have demonstrated reductions of 10^4 to 10^5 fibers/liter. At all milling facilities surveyed, reduction by at least a factor of 10 is realized, but the most common reduction factors range from 10^3 to 10^4 fibers/liters. Examination of these treatment systems indicates several factors in common: high initial suspended solids loading, effective removal of suspended solids, large systems or systems with long residence times, and/or the presence of additional settling or polishing ponds. Comparison of screen sampling data with verification sampling data at some facilities suggests that the asbestiform fiber content of the wastewater may be quite variable from time to time.

Seven mine water treatment systems exhibited two common characteristics: (1) generally low total-fiber and chrysotile asbestos counts, and (2) low to no removal of the fibers in their treatment systems. This can be explained by two factors: first, fibers tend to be liberated by milling processes, as compared to mining activities alone; and, second, mine waste streams tend to have considerably lower suspended-solids values than mill waste streams. Because of this, there is less opportunity for interaction between the fibrous particles and the suspended solids and their simultaneous removal by subsequent settling.

Ion Exchange

Ion exchange is basically a process for transfer of various ionic species from a liquid to a fixed media. Ions in the fixed media are exchanged for soluble ionic species in the wastewater. Cationic, anionic, and chelating ion exchange media are available and may be either solid or liquid. Solid ion exchangers are generally available in granular, membrane, and bead forms (ion exchange resins) and may be employed in upflow or downflow beds or column, in agitated baskets, or in cocurrent or countercurrent flow modes. Liquid ion exchangers are usually employed in equipment similar to that employed in solvent-extraction operations (pulsed columns, mixed settlers, rotating-disc columns, etc.). In practice, solid resins are probably more likely candidates for end-of-pipe wastewater treatment, while either liquid or solid ion exchangers may, potentially be utilized in internal process streams.

Individual ion exchange systems do not generally exhibit equal affinity or capacity for all ionic species (cationic or anionic) and, then may not be suited for broad-spectrum removal schemes in wastewater treatment. Their behavior and performance are usually dependent on pH, temperature, and concentration, and the highest removal efficiencies are generally observed for polyvalent ions. In wastewater treatment, some pretreatment or preconditioning of wastes to reduce suspended solid concentrations and other parameters is likely to be necessary.

Progress in the development of specific ion exchange resins and techniques for their application has made the process attractive for a wide variety of industrial applications in addition to water softening and deionization. It has been used extensively in hydrometallurgy, particularly in the uranium industry, and in wastewater treatment (where it often has the advantage of allowing recovery of marketable products). This is facilitated by the requirements for periodic stripping or regeneration of ionic exchangers.

Disadvantages of using ion exchange in treatment of mining and milling wastewater are relatively high costs, somewhat limited resin capacity, and insufficient specificity (especially, in cationic exchange resins for some applications). Also, regeneration produces a waste, and its subsequent treatment must be considered.

For recovery of specific ions or groups of ions (e.g., divalent heavy-metal cations, or metal anions such as molybdate, vanadate, and chromate), ion exchange is applicable to a much broader range of solutions. This use is typified by the recovery of uranium from ore leaching solutions using strongly basic anion exchange resin. Additional examples are the commercial reclamation of chromate plating and anodizing solutions, and the recovery of copper and zinc from rayon-production wastewaters. Chromate plating and anodizing wastes have been purified and reclaimed by ion exchange on a commercial scale for some time, yielding economic as well as environmental benefits. In tests, chromate solutions containing levels in excess of 10 mg/l chromate, treated by ion echange at practical resin loading values over a large number of loading/elution (regeneration) cycles, consistently produced an effluent containing no more than 0.03 mg/l of chromate.

High concentrations of ions other than those to be recovered may interfere with practical removal. Calcium ions, for example, are generally collected along with the divalent heavy metal cations of copper, zinc, lead, etc. High calcium ion concentrations, therefore, may make ion exchange removal of divalent heavy metal ions impractical by causing rapid loading of resins and necessitating unmanageably large resin inventories and/or very frequent elution steps.

Less difficulty of this type is experienced with anion exchange. Available resins have fairly high selectivity against the common anions, such as Cl- and SO_4^{-2}. Anions adsorbed along with uranium include vanadate, molybdate, ferric sulfate anionic complexes, chlorate, cobalticyanide, and polythionate anions. Some solutions containing molybdate prove difficult to elute and have caused problems.

Ion exchange resin beds may be fouled by particulates, precipitation within the beds, oils and greases, and biological growth. Pretreatment of water, as discussed earlier, is therefore commonly required for successful operation. Generally, feed water is required to be treated by coagulation and filtration for removal of iron and manganese, CO2, H2S, bacteria and algae, and hardness. Since there is some latitude in selection of the ions that are exchanged for the contaminants that are removed, post treatment may or may not be required.

In many cases, calcium is present in mining and milling wastewater in appreciably greater concentrations than the heavy metal cations to be removed. Ion exchange, in those cases is expensive and little advantage is offered over lime precipitation. For the removal of anions, however, the relatively high costs of ion exchange equipment and resins may be offset partially or totally by the recovery of a marketable product. This has been demonstrated in the removal of uranium from mine water (Refer to "Treatability Studies, Uranium/Vanadium Mill 9401," in this section).

Removal of molybdate ion from ferroalloy ore milling wastewater has been investigated in a pilot plant study (Reference Historical Data, Molybdenum/Tungsten/Tin Mine/Mill 6102 in this section). Treating raw wastewater containing up to 24 mg/l of molybdenum, the pulsed-bed ion exchange pilot plant produced effluent consistently containing less than 2 mg/l. Continuous operation was achieved for extended periods of time, with results indicating profitable operation through sale of the recovered molybdenum and the procedure was put into full-scale operation. The application of this technique at any specific site depends on a complex set of factors, including resin loading achieved, pretreatment required, and the complexity of processing needed to produce a marketable product from eluant streams.

Radium 226 Removal

Radium 226 is a product of the radioactive decay of uranium. It occurs in both dissolved and insoluble forms and, in this industry, is found predominantly in wastewater resulting from uranium mining and milling. Two treatment techniques are used in this industry, and they represent state-of-the-art technology. They are barium chloride coprecipitation and ion exchange.

Barium Chloride Coprecipitation. Coprecipitation of radium with a barium salt (usually, barium chloride) has typically been used for radium removal from uranium mining and milling waste streams in the United States and Canada (References 46 and 47). The removal mechanism involves precipitation of dissolved radium as the sulfate in the first step which results in a residual concentration of dissolved radium at this stage of approximately 20 ug/l. The dissolved radium concentration is then further reduced by coprecipitation, whereby radium sulfate molecules are incorporated into nascent crystals of barium sulfate. Dissolved radium concentrations can be less than or equal to 1 to 3 picocuries (picograms)/liter (pCi/l). Effective settling is necessary for removal of coprecipitated radium.

Dosages of 10 to 300 mg Ba/liter are generally required, depending upon the characteristics of the waste stream. It has been reported that 0.03 mole/liter of sulfate is required for effective removal of radium (Reference 48).

The removal of radium by adsorption of barite (the mineral form of barium sulfate) has also been demonstrated in the laboratory. More than 90 percent of the radium in uranium mine or mill wastewater has been removed in this manner by passing the waste stream through barite in a packed column (References 49, 50 and 51).

A number of facilities in the domestic uranium mining and milling industry use the barium chloride coprecipitation process for removal of radium, and this technology was used as the basis for BPT effluent limitations. A summary of facilities which effectively employ this technology is presented in Table VIII-9.

Ion Exchange. At uranium Mill 9452, a unique mine-water treatment system exists which uses radium 226 ion exchange in addition to flocculation, barium chloride coprecipitation, settling, and uranium ion exchange. The mine water to be treated is pumped from an underground mine to a mixing tank, where flocculant is added. The water is then settled in two ponds in series, before barium chloride is added. After barium chloride addition, the water is mixed and flows to two additional settling ponds (also in series). The decant from the final pond is acidified before it proceeds to the uranium ion exchange system. The uranium ion exchange column effluent is pumped to the radium 226 ion exchange system. After treatment for removal of radium 226, the final effluent is pumped to a holding tank for either recycle to the mill or discharge.

The total treatment system at Mine/Mill 9452 is capable of removing radium 226 from levels of 955 picocuries/liter (total) and 93.4 picocuries/liter (dissolved) to 7.18 picocuries/liter (total) and less than 1 picocurie/liter (dissolved). This performance represents 99.2 percent removal of total radium 226 and greater than 99 percent removal of dissolved radium 226.

Ammonia Stripping

High concentrations of ammonia in facility wastewater can be effectively removed by air stripping processes. In this mass transfer process, air and water are contacted in a packed or wet column. Water is sprayed from the top of the column and allowed to trickle down the wood or plastic media in packed columns, or fall as droplets in wet columns. Air is conducted in a countercurrent mode (from bottom to top of column) or a cross-flow mode (entering from the sides, rising and venting from the top of the column) through the system by one or more fans or blowers.

The efficiency of ammonia removal by this system depends on pH, temperature, gas-to-liquid flow ratio, ammonia concentration, and turbulence of flow at the gas-liquid interface. Strippers are operated at a pH of 10.8 to 11.5, which reduces the concentration ammonia (NH_3). Proper design of the stripping unit considers ammonia concentration and temperature to determine column sizing and gas/liquid flow rates.

Advantages of this system include its simplicity of operation and control. Some disadvantages are inefficiency at low temperatures (including freezeups at temperatures below 0 C), and formation of calcium carbonate scale on tower packing material (due to lime addition necessary for pH elevation). A further environmental consideration is the quantity of ammonia gas discharged to the atmosphere and its eventual impact on the concentration of ammonia in rainfall.

A variation of the ammonia stripping process, which is currently in its developmental stages, is a closed-loop system. This system recovers the stripped ammonia gas by absorption into a low pH liquid. The gas (initially air) passes from the stripping unit to an absorption unit where its ammonia content is reduced, and then returns to the stripping unit in a continuous closed loop operation. This type of system allows for recovery of the stripped ammonia and recycling of the absorption liquid in a second closed loop. Thus, the system avoids discharge of ammonia to water supplies as well as to the atmosphere. Also, since the equilibrium condition for the gas stream in a closed system is low in carbon dioxide, the problem of calcium carbonate deposits on the stripping media is avoided. Further description of these processes can be found in References 52 and 53.

Ammonia used in a solvent extraction and precipitation operation at one milling site is removed from the mill waste stream by air stripping. The countercurrent flow air stripper used at this plant operates with a pH of 11 to 11.7 and an air/liquid flow ratio of 0.83 m^3 of air per liter of water (110 ft^3 of air per gallon of water). Seventy-five percent removal of ammonia is achieved, reducing total nitrogen levels for the mill effluent to less than five mg/l, two mg/l of which is in the form of nitrates. Ammonia may also be removed from waste streams through oxidation to nitrate. Aeration will accomplish this oxidation slowly, and ozonation of chemical oxidants will do it quickly. However, these procedures are less desirable because the nitrogen still enters the receiving stream as nitrate, a cause of eutrophication.

PILOT AND BENCH SCALE TREATMENT STUDIES

Numerous pilot- and bench-scale wastewater treatment studies have been performed throughout the ore mining and dressing industry. These treatment studies were conducted to determine the following: the effects of combining various waste streams on wastewater treatability; the feasibility of employing a unit treatment process or system for removal of specific pollutants; the effluent quality attainable with a unit treatment process or system; the optimal operating conditions of a treatment process; and the engineering design parameters and the economics of building, operating, and maintaining a unit process or treatment facility. The studies were conducted by industry, the University of Denver, and EPA.

Copper Mine/Mill 2120

At copper Mine/Mill 2120, a bench-scale study of pH adjustment and settling was conducted by the company to determine the effects of combined treatment of water from an underground mine, barren leach water, and mill tailings. The individual and combined wastewater streams were treated with milk of lime to pH values ranging from 6 to 11 and allowed to settle for 20 minutes. The analytical results presented in Tables VIII-10 through VIII-13 demonstrate the heavy metals removal attained. Metals removal in all experiments were similar except for zinc, which was much more effectively removed by combined treatment of the wastewater.

The solids produced by treatment of the mine water or barren leach water were observed to be light and easily resuspended by turbulence caused by wind action. However, when these waters were treated in combination with tailings, the solids which settled were more dense and not as easily resuspended by wind generated turbulence. This led to the conclusion that the heavy metal precipitates produced by lime treatment of the combined wastewater streams are stabilized by the mill tailings.

Molybdenum Mine/Mill 6102

Molybdenum Mine/Mill 6102, which has historically discharged wastewater from its tailing pond system intermittently, has performed extensive bench- and pilot-scale evaluations of techniques for improving the quality of the wastewater discharged. In addition to conventional precipitation technology the following methods have been evaluated: ion exchange, alkaline chlorination, ozonation, and electrocoagulation flotation.

Molybdenum recovery by ion exchange was evaluated in an extensive pilot-plant study. This mill recycles water extensively and high levels of molybdenum, on the order of 20 mg/l, are commonly found in the discharge. Treatment of mill water in a pilot-scale, pulsed-bed, counter-flow ion exchange unit achieved substantial reductions in molybdenum concentrations, as demonstrated by the summarized results below.

Test Date (1975)	Mo(mg/l)		
	Feed	Effluent	Eluate*
7/24 and 7/25	20.5	1.18	16,140
7/28 and 7/29	23.0	0.91	16,045
7/29 and 7/30	22.4	1.38	16,568
7/31 and 8/1	24.4	1.76	18,090
8/1 and 8/2	19.5	1.14	12,930
8/5 and 8/6	22.0	1.38	17,484
Average	22.0	1.29	16,230

*Pregnant recovery fluid, see glossary.

For the period studied, service time was 41 minutes, resin-pulse volume averaged 1.73 liters (1.83 quarts), and flow-rate feed was 121 to 125 liters (32 to 33 gallons) per minute. Effluent concentrations of molybdenum were consistently below 2 mg/l. The high concentrations achieved in the ion exchange eluate allow economical recovery of the molybdenum, defraying a substantial fraction (or possibly all) of the costs of the ion exchange operation. On the basis of pilot-plant testing results, a decision was made to install a full-scale ion exchange unit.

Laboratory tests of precipitation technology at this site indicate that, at the low effluent temperatures which prevail, conventional precipitation technology would not be effective in removing heavy metals at retention times considered economical. Electrocoagulation flotation was evaluated as an alternative, and the pilot-scale unit was run to define optimum operating conditions and performance capabilities. Performance achieved at various operating pH levels is summarized below:

Parameter	Concentration (mg/l)			
	Feed	Effluent a	Effluent b	Effluent c
pH (units)	--	8.5	9.2	10.4
Iron	25 to 35	1.9	0.6	0.8
Manganese	6.3 to 6.6	1.6	0.5	0.1
Zinc	1.4 to 1.6	0.1	0.04	0.04
Copper	0.59 to 0.74	0.15	0.10	0.09
Cadmium	0.03 to 0.04	0.01	0.02	0.01
Cyanide	0.22 to 0.33	0.13	0.07	0.06

Cyanide destruction and removal techniques were also evaluated in conjunction with the electrocoagulation flotation pilot-plant. Removal by ferric hydroxide sorption was found to be ineffective. Ozonation did not consistently reduce cyanide to the desired levels; however, substantial reductions were achieved at elevated values of pH. Chlorination using sodium hypochlorite in excess (by a factor of 40) was found to be effective in reducing cyanide concentrations to less than or equal to 0.02 mg/l.

On the basis of pilot-plant test results, it was decided to install full-scale electrocoagulation flotation treatment, augmented by alkaline chlorination using sodium hypochlorite (generated on-site by electrolysis of sodium chloride). The treatment system treats a continuous bleed stream, with a capacity of 126 liters per second (2,000 gallons per minute), from the mill water system. Post treatment is planned, as required, to provide additional retention time for cyanide decomposition and to decompose chlorine residuals, probably by addition of sodium sulfide.

Actual performance capabilities of the full-scale system, which has been on-line at this facility since July 1978, are presented later in this section under the subheading Historical Data Summaries.

Lead/Zinc Mine/Mill/Smelter/Refinery 3107

Facility 3107, a lead/zinc mine/mill/smelter/refinery complex, has recently been investigating the feasibility of additional treatment using filtration. A pilot-scale pressure filtration unit is treating 9.5 to 31.5 liters per second (150 to 500 gallons per minute) of treatment system effluent using granulated slag as the filtration medium. Full-scale designs currently under consideration will provide a maximum effluent total suspended solids concentration of 5 mg/l with 100 percent reliability (industry report).

Lead/Zinc Mine/Mill 3144

Laboratory and pilot plant studies were conducted at lead/zinc Mine/Mill 3144 in 1973 to define an effective treatment for the destruction of cyanide. Preliminary laboratory tests were conducted using calcium hypochlorite as an oxidizing agent. Although this agent effectively destroyed cyanide contained in the mill wastewater, the use of hypochlorite in a full-scale operation was deemed inefficient and uneconomical (Reference 17). As a result, a second series of tests was conducted using chlorine gas as the oxidizing agent. Based on the results of these tests, construction of a full-scale chlorination plant was initiated in mid-August 1973. Startup operation of the full scale plant began in December 1973. Monitoring data indicate that the full scale plant effectively reduces cyanide (total) from an average of 68.3 mg/l in the raw waste to an average of 0.13 mg/l in the treated effluent. The design and operating characteristics of the full-scale plant have been previously described in Technique Description, Cyanide Treatment earlier in this section.

Canadian Mine Drainage Study

During 1973 and 1974, a pilot treatment plant was operated at a mill located in New Brunswick, Canada, to demonstrate the treatability of base-metal mine water discharge using conventional treatment technology and to define the factors critical to the optimization of treatment. Treated effluent polishing techniques were also evaluated and a final report of the project was published (Reference 54). Several earlier reports described the treatment plant design, optimization, and capabilities and the development of flocculant addition and sludge handling and dewatering methods (References 55, 56, 57, and 58).

The pilot-plant treatment included provisions for two-stage lime addition, coagulation, mechanical clarification, and sludge recycle. Effluent polishing techniques employed included additional settling or sand filtration. Treatments of three acidic, metal-bearing mine drainages were evaluated in the pilot-plant. The characteristics of these three mine drainages are shown in Table VIII-14. As indicated, the individual mine drainages greatly differ in acidity and total metal content. Results of treatment studies are summarized in Table VIII-15.

The principal findings of this pilot-plant project are summarized as follows:

1. The following metal levels were attained as clarifier overflow concentrations, on an average basis, for the various drainages treated during periods of steady operation:

Metal	Concentrations (mg/l)	
	Extractable (total)	Dissolved
Pb	0.25	0.24
Zn	0.37	0.26
Cu	0.05	0.04
Fe	0.28	0.22

2. Polishing of the clarifier overflow by sand filtration and bucket settling further reduced the above extractable total metal levels. Levels attained on an averaged basis were:

Metal	Concentration (mg/l)
	Extractable (total)
Pb	0.12
Zn	0.19
Cu	0.04
Fe	0.17

3. The initial acidity and total metal concentrations had little effect on the final effluent quality, but greatly influenced the volume and density of sludge produced; these factors affected the quantity of neutralizing reagent required.

4. Optimization of the coagulant (polymer) addition by experiments run on the wastewater in question was determined to be the process most critical to obtaining low concentrations of metals in the clarifier overflow. The beneficial effect of polymer addition was clearly demonstrated in the treatment of mine 3 drainage, where polymer additions increased settling rates fourfold (1.8 to 7.4 m/hr, equal to 5.9 to 24.3 ft/hr) and reduced the total metal concentrations of the clarifier overflow sixfold. In the case of mine 2 drainage, polymer addition reduced metal concentrations by an additional 30 to 50 percent and increased settling rates fivefold (0.45 to 2.4 m/hr, equal to 1.5 to 7.9 ft/hr) during once-through operation of the clarifier (i.e., no sludge recycle).

5. No performance advantages were found in two-stage lime neutralization compared to single-stage lime neutralization. The sensitivity of the process was found to be a function of solid/liquid separation and not pH, provided that the pH was maintained within one pH unit of the optimum.

6. Sand filtration and quiescent settling were shown to be effective methods of further reducing metal values in clarifier overflow and reducing the variability in these levels.

University of Denver Mine-Drainage Study

The University of Denver, in cooperation with EPA and the State of Colorado Department of Health and the Department of Game, Fish, and Parks, has conducted field experiments to evaluate the treatability of metal-bearing mine drainage from mines in the highly pyritic districts of the San Juan Mountains of southwestern Colorado (References 35 and 39). Charactersitics of this mine drainage are tabulated below:

Parameter	Concentration (mg/l)
pH (units)	2.6 to 3.1
Fe	336 to 800
Cu	51.6 to 128
Mn	4.52 to 19.0
Al	20.8 to 62.5
Zn	122 to 294
Pb	0.04 to 0.50
Ni	0.19 to 0.51
As	6.01 to 22.0
Cd	0.44 to 1.0
Sulfate	1,400 to 3,820

The study was conducted specifically to evaluate the capabilities of the treatment scheme depicted in Figure VIII-2. This treatment consisted of a two-stage process of chemical addition to the mine drainage, followed by settling. The first stage consisted of lime addition, and the second stage involved sulfide addition. The pH was very easy to control with the first stage achieving pH 5.0, and the second stage achieving pH 6.5 (the ambient pH of the region). A second finding of the field studies was that moderate, wind-induced turbulence in the settling pond would maintain hydroxide flocs in suspension, while the sulfide precipitates settled immediately. During the field experiment, pH was varied in the two stages of treatment. The results are tabulated below:

	Exp. 1	Exp. 2	Exp. 3	Exp. 4	Exp. 5
pH (Stage 1)*	5.0	5.5	5.0	5.0	5.0
pH (Stage 2)*	6.4	6.4	5.5	5.6	6.5
Fe (total)	ND	ND	ND	ND	ND
Zn	12.7	0.3	30.0	30.0	0.2
Mn	6.4	0.5	6.8	7.1	0.4
Cu	ND	ND	LT 0.5	LT 0.3	ND
Al	ND	ND	ND	ND	ND
Ni	LT 0.13	0.13	0.29	0.19	0.13
Cr	ND	ND	ND	ND	ND

*Value in pH units
LT = less than or equal to
ND = below detection limit

The experiment 5 results tabulated above, were reported to define the standard design condition, which was held at steady-state for an extended period. For this condition, the following effluent concentrations of additional metals were attained:

Hg	0 mg/l
Cd	0.008 mg/l
As	0 mg/l

Lead/Zinc/Gold Mine 4102

Drainage from lead/zinc/gold Mine 4102 enters a precipitous avalanche area which is too small for construction of settling ponds of adequate size for conventional pH adjustment and settling of mine water. As a result, research has been conducted at this facility to design a satisfactory treatment system.

Various techniques, such as adsorption, reverse osmosis, and ion exchange, were initially considered for treatment of the mine drainage. However, chemical precipitation (conventional lime precipitation) was ultimately chosen as the treatment process. As indicated in Table VIII-16, this method has been demonstrated to effectively precipitate dissolved metals present in the mine water at elevated pH (i.e., pH 9.2).

After conventional lime treatment was selected as the alternative, an evaluation of technologies for removal of suspended solids was initiated. On the basis of the test results, the EnviroClear and Lamella Gravity Settler techniques (commercial package treatment systems) were determined to be efficient and practical means of treatment which require a minimum amount of space.

A number of sludge dewatering or sludge thickening methods were investigated, including conventional sludge filtration (both the drum filter and the frame filter press); centrifugation; the Parkson Corporation's "Magnum Press"; Carborundum Company's "New Sludge Filtering System"; Aerodyne Corporation's "Filtration Cylinder"; and Enviro-Clear Company's "New Belt Filter." Sludge recycle and use of coagulants were also considered as methods to enhance settling and sludge dewatering. Although some of the methods investigated were technologically feasible, no final choice of a sludge dewatering or sludge-handling method was identified as preferable on an economic basis.

Lead/Zinc Mine 3113

Bench scale studies were conducted by mine personnel at lead/zinc Mine 3113 in 1975 through 1976 to evaluate the effectiveness of a proposed treatment system to handle 6,400 m^3 (1.7 million gallons) of mine water drainage per day which is discharged without treatment. The basic treatment scheme investigated consisted of lime addition to pH 10 to 11, followed by sedimentation. Sludge thickening and polyelectrolyte addition were also evaluated. The results of these tests are presented in Table VIII-17.

In subsequent bench-scale tests, mine-water samples were shipped to the manufacturer of a high rate settling device (Lamella Gravity Settler) to evaluate the effectiveness of this device when used in conjunction with lime and polyelectrolyte. The best overflow quality and sludge dewatering properties were attained with 1.0 to 1.8 mg/l polymer. Lime requirements were reduced by 15 percent by presettling prior to lime addition.

Based on the results of the treatment studies, a full-scale mine water treatment scheme was developed. Mine water drainage would be pumped to a lined holding lagoon with a theoretical retention time of 24 hours (value = 6,400 m^3 or 1.7 million gallons). Mine water would be pumped from the holding lagoon to a tank, where lime would be added to raise the pH to the range of 10.5 to 11.0. Overflow from the lagoon would flow by gravity to a flash-mix tank, where coagulant would be added to the stream prior to passage to a pair of Lamella Gravity Settlers (in parallel). Overflow from the settling units would flow to a polishing lagoon with a theoretical retention time of approximately 6 to 9 hours with sulfuric acid neutralization prior to discharge.

EPA Treatability Studies

In August 1978, comprehensive studies of the treatability of wastewater streams from ore mining and milling facilities were initiated by Calspan Corporation under contract to EPA (Contract 68-01-4845). The primary purpose of this program was to delineate the capabilities of BAT alternative treatment technologies for mine and mill waters, technologies for the treatment of uranium mill wastewater, and to expand the data for technologies for which little or no empirical information was available. In addition, the operating conditions were varied at each site for the pilot-scale system used in the studies. This was done to clarify engineering and economic considerations associated with designing and costing full-scale versions of the treatment schemes investigated.

The studies were performed at seven ore mining and milling facilities and the results are summarized in this document (Refer to Table VIII-18). A detailed discussion of these studies, their analytical results, and the experimental designs and procedures is presented in Reference 59. A discussion of two treatability studies at Facility 2122 is presented in this section, under the heading ADDITIONAL EPA TREATABIITY STUDIES.

All EPA-sponsored pilot scale treatability studies were conducted on-site using a 2.4-meter (8 foot) by 12.2-meter (40 foot) semitrailer designed specifically for performance of pilot- and bench-scale wastewater treatment studies in the field. This mobile treatment plant provides the following unit processes, either individually or in combination, on a pilot-scale: flow equalization, primary sedimentation, secondary sedimentation, pH adjustment, chemical addition (polymer, lime, ferrous sulfate, sodium hydroxide, barium chloride, sodium hypochlorite, and others) coagulation, granular media pressure filtration, ozonation, aeration, alkaline chlorination, ultrafiltration, flotation, ion exchange and reverse osmosis. A schematic diagram of the basic system configurations used are presented in Figures VIII-3, VIII-4 and VIII-5.

Fifteen parameters, (pH, total suspended solids, and 13 toxic metals) were monitored at all sites. Additional parameters such as iron, aluminum, molybdenum, vanadium, radium 226, uranium, phenols, and cyanide were monitored at appropriate sites. Results of the testing at various facilities follow.

Lead/Zinc Mine/Mill 3121. At this facility, lead/zinc ore is mined from an underground mine and concentrated in a mill by the froth flotation process. Mine drainage is combined with mill tailings for treatment in a tailing pond. A coagulant (polymer) is added to the combined waste stream to improve settling in the tailing pond. However, the tailing pond provides limited retention time, and the tailing pond decant generally contains relatively high concentrations of metals. Therefore, the pilot-scale treatment schemes investigated at this facility consisted of add-on or polishing technologies for improved removal of metals from the tailing-pond effluent. Pilot-scale unit processes used included lime addition for pH adjustment, coagulant (polymer) addition as a settling and filtration aid, secondary settling, and dual-media filtration.

The study at Mine/Mill 3121 was performed in two segments, the first segment during warm weather (August), and the second segment during cold weather (March). This scheduling was deliberate since cyanide and copper concentrations in the tailing pond decant at this facility are generally much higher during the cold months than during the warm months. A major goal of the second segment of this study was to determine the removal of copper by effluent polishing techniques when relatively high concentrations of both copper and cyanide were present. In addition, the capabilities of alkaline chlorination and ozonation for destruction of cyanide in the tailing pond decant were studied during March. For reasons which will be discussed, the cyanide destruction studies could not be completed.

A characterization of the wastewater influent (i.e., the tailing pond decant) sent to the pilot-scale treatment system during the period of study is presented in Tables VIII-19 and VIII-20. As illustrated, pH, TSS, and total metals concentrations varied over a wide range during the August study. This variability appeared to be related to the schedule of mill operation (Refer to Table VIII-21). During periods when the mill was not operating, only mine water was being discharged into the tailing pond. (However, this facility does not use lime treatment; alkaline mill water provides pH adjustment for mine water). During the March study, the concentrations of the parameters of interest in the decant were generally much higher and less variable than during August.

Two basic experimental designs were employed to investigate metal removal by effluent polishing. Initially, direct filtration of the tailing pond decant was investigated. Subsequently, a second set of experiments was conducted to determine the improvement in metals removal attained by lime addition, coagulant (polymer) addition, and settling prior to dual-media filtration.

A summary of the treated effluent concentrations attained is presented in Tables VIII-22 (August study) and VIII-23 (March study). Results of experiments to evaluate cyanide destruction by ozonation are presented in Table VIII-24. Conclusions and observations made on the basis of these results are summarized below:

1. During the August study the metal removal efficiency of filtration was found to be dependent on the pH maintained in the tailing pond system. Metal removal efficiency improved with increasing pH.

2. Filtration consistently reduced the total suspended solids concentrations of tailing-pond decant to 1 mg/l or less during the August study.

3. Also during the August study, filtration consistently reduced total metal concentrations to levels well below BPT limitations when the pH of the waste stream was in the range of 7.7 to 11.3. Zinc concentrations in excess of 0.5 mg/l were observed in filtrates when the pH was below 7.7.

4. The results of this study demonstrated the chemical addition/settle/filtration treatment scheme to be very effective for removal of TSS and metals. Furthermore, this treatment system was 30 to 90 percent more efficient in the removal of TSS and metals than filtration alone. Total copper was reduced from an initial concentration of 0.15 to 0.19 mg/l to a final concentration of 0.12 to 0.16 mg/l attained by filtration alone.

5. While the results of these studies indicate that copper can be removed from the wastewater of Mine/Mill 3121, some additional observations need to be made. First, the concentration of dissolved copper in the tailing pond decant during the August study ranged from 0.010 to 0.040 mg/l. During the March study the dissolved copper concentration was 0.010 to 0.050 mg/l. These concentrations are low relative to the 30-day average dissolved copper concentrations, typically 0.08 to 0.22 mg/l, reported by Mine/Mill 3121 for NPDES monitoring purposes. Second, the concentration of cyanide in the tailing pond decant during August was 0.07 to 0.08 mg/l. During March the concentration of cyanide ranged from 0.04 to 0.125 mg/l. The cyanide concentration during August was typical of the warm weather months, but the concentrations for March were much lower than normal for the time of year (see Table VIII-20). On the basis of these low dissolved copper and cyanide concentrations, it does not appear that a copper cyanide complex could have been present at any significant concentration during either of the two treatability studies conducted at Mine/Mill 3121. Therefore, there is no evidence that copper can not be removed from this facility.

6. Results of cyanide destruction by ozonation indicate the greatest degree of cyanide destruction occurred at the highest ozone dosage rate. Even at a 10:1 ratio of ozone to cyanide the efficiency of cyanide destruction was only 43 percent. This is due to the initial low concentration of cyanide (i.e., 0.115 mg/l) and the problems involved in the mass transfer of small amounts of ozone gas and contact with cyanide in a dilute solution.

7. During the March study the cyanide concentration in the tailing pond decant decreased to 0.04 mg/l before the

experiments to evaluate the destruction of cyanide by alkaline chlorination and ozonation could be completed. With an initial total cyanide concentration of only 0.04 mg/l it was considered to be impractical to continue these experiments. Therefore, only limited results for cyanide destruction by ozonation were obtained and none were obtained for alkaline chlorination.

Lead/Zinc Mine/Mill/Smelter/Refinery 3107. Wastewater streams generated from mining, milling, smelting, and refining activities at this lead/zinc complex are combined in a common impoundment pond, and the effluent from this pond is subsequently treated in a physical/chemical treatment plant by lime precipitation, aeration, flocculation, and clarification, in conjunction with high- density sludge recycle. Treated effluent from this facility is characterized as being alkaline with relatively high concentrations of zinc, cadmium, lead, and total suspended solids (refer to Table VIII-25).

The pilot-scale treatment schemes investigated at this facility focused on end-of-pipe polishing technologies for improved removal of suspended solids from the treated effluent. The unit processes investigated were dual-media granular pressure filtration and supplementary sedimentation. The use of polymers and flocculation as settling aids was also investigated.

A characterization of wastewater treated in the pilot-scale system is presented in Table VIII-26. It is evident from a review of this table that metals in this waste stream are components of the suspended solids, since the dissolved metal concentrations are very low relative to total metal concentrations.

A summary of results for the treatment schemes investigated is presented in Table VIII-27. Treatment efficiencies are reported only for BPT control parameters and other parameters present at significant levels in the raw wastewater.

Results of this study indicate that the suspended solids (especially, the metal hydroxide flocs) in the effluent from the physical/chemical treatment plant are filterable and not subject to shear in the filters. Total suspended solids were consistently removed to less than 1 mg/l by all three filter configurations investigated and over the range of hydraulic loadings employed (i.e., 117 to 880 m^3/m^2/day, 2 to 15 gpm/ft^2). Correspondingly, metals were effectively removed by filtration.

Secondary settling reduced suspended solids by 81 percent from an average of 16 mg/l to 3 mg/l, with metal removals ranging from 38 percent (an average of 0.13 mg/l to 0.18 mg/l) for lead to 72 percent (an average of 2.9 mg/l to 0.79 mg/l) zinc (Table VIII-27). A theoretical retention time of 11 hours was employed for secondary settling experiments. The effluent quality produced by secondary settling was not as good as that produced by dual-media filtration, probably due to the poor settling characteristics of metal hydroxides (especially, zinc hydroxide).

A non-ionic polymer did not appear to enhance the settleability or filterability of the wastewater treated. However, sufficient time was not available for process optimization (i.e., polymer and dosage selection, flocculation time, agitation intensity during flocculation, etc.).

Lead/Zinc Mine 3113. Drainage from this lead/zinc mine flows primarily from extensive inactive mine workings. Occasionally mine water from an active mine is discharged via the mine drainage system. Mine 3113 drainage is characterized as acidic, with high concentrations of heavy metals, especially iron and zinc (refer to Table VIII-28). The experiments conducted at this site were to determine the quality of effluent which could be attained by treatment of the mine drainage with lime precipitation, flocculation, aeration, and mixed media filtration processes. At present, this drainage is discharged without treatment.

The character of the mine water treated during the period of study is presented in Table VIII-29. It should be noted that the pH is low; Cd, Cu, and Zn concentrations are practically all dissolved; and Fe is less than one half dissolved. Results of the pilot-scale treatability studies are summarized in Table VIII-30.

Experimental treatment systems E, G, and I in Table VIII-30 were designed to investigate the efficiency of lime addition, aeration, polymer addition, flocculation, and settling at various pHs. With the exception of zinc, the efficiencies of metals removal were practically identical and independent of the pH. However, in the case of zinc, a relationship was found between the efficiency of removal and pH, and the greatest removal of zinc occurred at the highest pH (10.5). In contrast to the improved efficiency of zinc removal, the total suspended solids concentration increased with increasing pH. This was the result of the increased lime dosages required to attain the higher pH levels.

Experimental systems identified as A and C in Table VIII-30 were designed to investigate anticipated improvements in treatment efficiency by using aeration to oxidize ferrous (Fe^{+2}) ion to the ferric (Fe^{+3}) state. A ferric hydroxide precipitate is formed (in system C); however, only slightly improved iron removal (4.8 to 4.0 ug/l) was observed. No improvement in TSS or other toxic metals removal was observed. Heavy reddish-brown sediments (yellowboy) were observed in the mine-drainage discharge ditch, indicating that the iron present was mostly oxidized.

Experimental systems identified as C and E in Table VIII-30 were used to investigate the extent to which the treatment efficiency of the basic lime and settle treatment system could be improved by incorporating polymer addition and flocculation. Review of the experimental results demonstrates that polymer addition, followed by flocculation, greatly improved the capabilities of the basic lime and settle system. Most notable was the threefold improvement in removal efficiency of total suspended solids, zinc, and iron.

A dual-media, granular filtration step was used with all systems as a final polishing step (systems D, F, H, and J). The incremental improvements in removal of total suspended solids and total metals resulting from filtration are also represented in Table VIII-30. Results indicate filtration is an effective polishing treatment showing significant (14 to 5 mg/l) improvements in TSS in all cases and general improvement in metals concentrations.

On the basis of the experimental results, the treatment scheme producing optimum removals of suspended solids and heavy metals from acid mine drainage at Mine 3113 consisted of adjustment of pH to 10.5 with lime, flocculant addition, flocculation, sedimentation, and filtration (experimental system J in Table IX-30). Other, less rigorous treatment schemes with lower lime dosages and without filtration would not reliably produce the excellent effluent quality attained with system J.

Aluminum Mine 5102. At this site, bauxite is mined by open-pit methods. Watewater (approximately 17,000 m³, or 4.5 million gallons, per day) emanates as runoff and as drainage from the open-pit mine. This wastewater is generally characterized as acidic (with a pH of 2.2 to 3.0), with total iron and aluminum concentrations in the range of 50 to 150 mg/l and 50 to 200 mg/l, respectively. The treatment system used for this mine water consists of lime addition and sedimentation in a multiple pond system.

The character of treated mine water (influent to pilot-scale treatment system) during the period of study is presented in Table VIII-31. As indicated, concentrations of the 13 toxic pollutant metals were found to be either below detection limits or only slightly above detection limits of atomic adsorption spectrophotometric analysis. Other parameters (TSS, iron, and aluminum) were present at higher concentrations, but were well below BPT limitations.

The basic pilot-scale unit treatment processes investigated at Mine 5102 consisted of lime addition, aeration, polymer addition, flocculation, sedimentation, and dual-media filtration. Results of the treatability studies are summarized in Table VIII-32. These results are of limited value because the waste stream being treated was already of a very high quality (low pollutant concentrations).

Findings of the treatability studies at Mine 5102 are summarized below:

1. Elevated pH, in the range of 8.2 to 10.7, did not significantly improve or degrade removal efficiency of aluminum or iron, as illustrated by the results shown in Table VIII-32. This is to be expected from the low dissolved metal concentrations in the mine water influent to the pilot-scale treatment units (Table VIII-31).

2. The use of a polymer improved settling performance during lime addition experiments. species, i.e., arsenate ($AsO4^{-3}$), molybate (MoO_4^{-2}), selenite)SeO_3^{-2}), and vanadates (HVO_4^{-2}, VO_4^{-3}, etc.). These metal species are highly soluble at alkaline pH and cannot be removed by precipitation as hydroxides or carbonates.

3. Filtration consistently produced effluent total suspended solids concentrations of 1 mg/l or less. Total aluminum and iron concentrations were equal to the corresponding dissolved metal concentrations, indicating essentially complete removal of particulate metal compounds. The use of hydraulic loadings of 117 to 880 m^3/m^2/day (2 to 15 gpm/ft^2) and effective filter media sizes over the ranges of 0.35 mm to 0.7 mm, respectively, did not significantly alter the quality of effluent attained.

Uranium Mill 9402. This mill, like all domestic uranium ore mills, is located in an arid region and attains zero point discharge of wastewater. This is accomplished by recycling and using evaporation ponds. It was selected as a wastewater treatability site for several reasons. It is possible that regulations will be developed to protect groundwater quality pursuant to other statutes. Such regulations could mandate the use of seepage control devices and practices which would eliminate loss of wastewater from tailing ponds by seepage. This could ultimately necessitate end-of-pipe discharges of wastewater at uranium mills presently attaining zero discharge. For these reasons, EPA found it desirable to investigate treating wastewater generated at uranium mills.

As discussed in Section III, two basic processes are employed at uranium mills, acid leaching and alkaline leaching. The pH in wastewater produced in an acid leach circuit are different from those in wastewater produced in an alkaline leach circuit. Therefore, uranium mills representative of both processes were selected. Mill 9402 was selected as a representative acid leach mill; Mill 9401, discussed later, is an alkaline leach mill.

The wastewater treatability study conducted at the uranium Mill 9402 was performed in two phases. The initial phase was performed during the period of 4 to 9 November 1978 and the second phase during the period of 3 to 12 December 1978. The basic pilot-scale treatment scheme employed during the initial phase consisted of lime precipitation (pH adjustment), aeration, flocculant (polymer) addition, barium chloride coprecipitation, flocculation, single-stage or two-stage settling, and mixed-media filtration. Reagent dosages and operating parameters employed during the initial phase of this study were chosen largely on the basis of previous laboratory studies conducted by the Australian Atomic Energy Commission (Reference 49) and by Calspan Corporation (Reference 59).

During all experimental runs the inclined settling tank used in the pilot plant was operated in a manner similar to a sludge blanket clarifier (see Figure VIII-6). However, during the initial phase of the study, the sludge produced was light and unconsolidated. For this reason, it was impossible to maintain a surface layer of clarified supernatant within the settling tank and the effluent contained high concentrations of total suspended solids. As a result, the total concentrations of certain metals in the effluent also tended to be high, although the dissolved concentrations were relatively low.

During the second phase of the study an attempt was made to improve this situation by incorporating sludge recycle and a metered sludge bleed into the pilot-scale system (see Figure VIII-4). This modification was added specifically to consolidate and thicken the sludge. Sufficient time was not available to optimize this process, but it was successful enough to allow continuous operation of the pilot plant. During the final experimental runs, the sludge recycle produced a noticeably thicker sludge and made it possible to maintain a 10 to 15-centimeter (4 to 6-inch) layer of clarified supernatant in the settling tank.

A summary of the physical/chemical characteristics of the raw wastewater (i.e., tailing pond seepage) at Mill 9402 which was used in the treatability studies is presented in Tables VIII-33 and VIII-34. Examination of these tables reveals that this wastewater is very acidic and contains very high concentrations of total dissolved solids, dissolved metals, and radium 226 (total and dissolved). A comparison of these two tables further indicates that the character of the wastewater during the second phase of the study (December) was somewhat different from the wastewater character during the initial phase of the study (November). Specifically, several parameters including TSS, Mo, Fe, Mn, and Al were present at much higher concentrations during the second phase of the study. The higher concentrations of these parameters were not found to have a readily apparent impact on the treatment system capabilities.

A review of the raw wastewater character at Mill 9402 demonstrates the metals present at high concentration to be Cu, Pb, Zn, Ni, Cr, V, Mo, Fe, Al, and Mn. Addition of lime to form metal hydroxide precipitates is known to be an effective removal mechanism for all of these metals except V and Mo (References 38, 60, 61 and 62). However, the high concentrations of Fe and Al in the wastewater suggested other possible removal mechanisms for these latter two metals. Hem (1977) has reported that the solubility of vanadium and molybdenum may be controlled by precipitation of iron vanadates and molybdates over the pH range of 3 to 9 for vanadate and 5.3 to 8.3 for molybdate (Reference 63). Michalovic et al (1977) conducted laboratory studies and reported that excess ferric hydroxide formed by oxidation and hydrolysis of ferrous sulfate was found to consistently yield vanadate (+5) concentrations of less than 4 mg/l (Reference 64). The removal mechanism proposed involved precipitation/coagulation as given below:

$$2Fe^{+3} + 3\ (VO_3)^{-1} + 3OH^{-} \text{ ---> } Fe(VO_3)^3 + Fe(OH)^3$$

The pH was adjusted by addition of lime and a final pH of 7 was reported to provide the best results. Similarly, Kunz, et. al. (1976) reported that the results of screening tests showed that ferrous sulfate provided the most efficient removal of vanadium anions (Reference 65). Concentrations of vanadium of less than 5 mg/l were attained when the pH was kept between 7.5 and 9 for V+4 precipitation and between about 6 and 10 for V+5 species. Again, the removal mechanism was thought to involve simultaneous precipitation of $Fe(VO_3)_2$ and Fe(OH). Similar removal mechanisms have been reported for Mo (Reference 66). However, the optimum pH for Mo removal using iron salts is much lower than required for V removal (i.e., about pH 3 to 4 for Mo). During laboratory studies significant removal of Mo has also been attained with aluminum hydroxide at a pH of about 4.5 (Reference 66).

On the basis of the wastewater (i.e., tailing pond seepage) characteristics and the literature summarized above, it was anticipated that the treatment scheme chosen for pilot scale testing would provide effective removal of most of the metals of concern. However, the removal of Mo and total dissolved solids (TDS) were expected to present some problems. The pilot scale experiments conducted were designed to investigate the effect of variable lime and barium chloride dosages on removal of metals, TDS, and coprecipitation of radium 226, respectively. A summary of the study results is presented in Table VIII-35.

Generally, pH values greater than the range 8.2 to 9 were required for optimum removal of TDS and most metals. However, optimum removal of molybdenum occurred at the lowest pH range investigated, pH 5.8 to 6.1, and improved removal of this metal would probably require operation at an even lower pH. A barium chloride dosage of 51 to 63 mg/l was required for optimum removal of radium 226.

The basic treatment scheme employed did not demonstrate effective removal of ammonia because it was not designed for removal of this parameter.

As described previously, the major operating problem encountered was maintaining control over suspended solids and sludge removal in the sedimentation unit. The metals which appeared to be the most sensitive to this problem were zinc, uranium, and radium 226.

Reduction of effluent TSS concentrations would also improve the total metal concentrations for all the metals subject to precipitation or coprecipitation removal mechanisms.

Uranium/Vanadium Mill 9401. This mill is located in an arid region of New Mexico. An alkaline leaching process is employed at this mill to selectively leach uranium and vanadium from ore. This facility achieves no end-of-pipe discharge of process wastewater by: (1) net evaporation (due to location in an arid region), (2) loss of water as seepage, and (3) recycle. The rationale for selection of this uranium mill as a treatability site was dicussed for uranium Mill 9402, which uses acid leach; Mill 9401 uses alkaline leach.

Clarified water from the tailing pond is passed through an ion exchange column prior to recycle to the mill leaching circuit. The purpose of the ion exchange unit is to recover solubilized uranium present in the recycle stream. This is apparently an economically feasible process for this mill, since the mill has continued to operate and recover uranium which would otherwise be lost by this approach.

Treatability experiments conducted on the water recycled from the tailing pond focused on removal/recovery of dissolved uranium and removal of other dissolved components, especially metals. As indicated in Table VIII-36, the metals of highest concentration (other than U) and, therefore, of interest are arsenic, molybdenum, radium 226, selenium, and vanadium. The highly alkaline and oxidized character of the mill wastewater and the existence of the metals in soluble form indicates their presence as anionic species, i.e., arsenate (AsO_4^{-3}), molybate (Mo_4^{-2}), selenite (SeO_3^{-2}), and vanadates (HVO_4^{-2}, VO_4^{-3}, etc.). These metals species are highly soluble at alkaline pH and cannot be removed by precipitation as hydroxides or carbonates.

Therefore, the treatability experiments conducted at this site focused on two basic treatment schemes. The first involved passing the wastewater through an ion exchange column containing amberlite IRA-430 resin to remove uranium. Ion exchange was followed by coprecipitation with ferrous sulfate, alum, or lime in conjunction with H_2SO_4, polymer addition and flocculation, and aeration. Results of preliminary bench scale experiments indicated that of the three chemical reagents investigated, (ferrous sulfate, alum and lime), ferrous sulfate provided the most effective removal of metals and was employed during all subsequent pilot scale experiments. Therefore, the basic pilot-scale treatment scheme consisted of ion exchange followed by ferrous sulfate addition/pH adjustment/aeration, barium chloride addition for coprecipitation of radium 226, polymer addition, flocculation, sedimentation, and dual media filtration. A schematic representation of the pilot-scale treatment system used is presented in Figure VIII-5.

The second treatment scheme investigated used the mixture, in varying proportions, of wastewater from an acid leach uranium mill (Mill 9402) and the alkaline leach wastewater of Mill 9401. Bench-scale treatment units were used.

Results of the ferrous sulfate/barium chloride coprecipitation system investigated at pilot-scale are presented in Table VIII-37. Results of the acid mill/alkaline mill wastewater admixture treatment scheme investigated at bench-scale are presented in Table VIII-38. A summary of the raw wastewater character (i.e., tailing pond recycle water) is presented in Table VIII-36.

The results summarized in Table VIII-38 indicate that ion exchange removed approximately 97 to 99 percent of the uranium present in the waste stream while 98 percent removal of radium 226 was attained by barium chloride coprecipitation with a $BaCl_2$ dosage of 15 to 60 mg/l. The effectiveness of removing vanadium, molybdenum, and selenium increased with decreasing pH. At pH 8.0, approximately 80 percent of the vanadium and 50 percent of the molybdenum and selenium were removed. The TDS concentration remained high (in excess of 20,000 ug/l) in the effluent because (1) the metals precipitated were at comparedly (compared to TDS) insignificant concentrations and (2) dissolved solids in the form of Fe (SO_4), $BaCl_2$ and polymer were added to the water as part of the treatment.

Because acid and alkaline leach uranium mills are sometimes located in close proximity, the mixture of wastewater from these two types of mills for neutralization and treatment may be a feasible alernative. Therefore, admixture experiments were conducted to investigate the degree of neutralization and the removal of molybdenum, selenium, and vanadium attained. These metals were of special interest since they are extremely difficult to remove from wastewater.

Enhanced metals removal was observed under all conditions studied. Optimum removal of metals was achieved at the highest ratio of acid to alkaline wastewater investigated (i.e., 5:3 ratio by volume). Even at a ratio of 5:4 acid to alkaline wastewater the removal efficiency of both Mo and V exceeded 97 percent. At admixture ratios of 5:4 and 5:3 by volume the final pH attained was 4.3 and 3.9, respectively. The amount of iron remaining in solution following admixture and the final pH suggests that a lime barium chloride addition treatment scheme would be very effective for subsequent treatment of the acid/alkaline wastewater mixture (see discussion of treatability study conducted at Mill 9402). Time did not permit investigation of lime/barium chloride precipitation, however.

It is notable that the admixture treatment scheme was the only scheme investigated which resulted in the effective removal of molybdenum. In view of the high cost required for neutralization of acid or alkaline uranium mill wastewater (if such treatment were ever required), admixture provides an additional advantage.

EPA-Sponsored Studies at Complex Facilities

During the month of September 1979, two pilot-scale treatability studies were conducted by Frontier Technical Associates, Inc. under contract to the EPA (Contract No. 68-01-5163). These studies were conducted to gather data on treatment at mine/mill/smelter/refinery complexes. The studies were conducted on-site using an EPA mobile laboratory truck and company owned, pilot scale treatability equipment. This equipment included a 90 gallon batch lime mix tank, dual media filter column, flow tray, 100-gallon filtrate holding tank, and associated pumps, piping, valves, and instrumentation. Figure VIII-7 is a schematic of the pilot-plant configuration.

The test operating conditions were varied at each site. Tests included combinations of pH adjustment by lime addition, secondary settling, dual media filtration, and dosing with hydrogen peroxide for cyanide treatment.

Samples were monitored for pH, total suspended solids, cyanide, phenols, and the 13 toxic metals.

Copper Mine/Mill/Smelter/Refinery 2122. The wastewater treatment plant at this facility treats the combined waste streams from two mills, a refinery (including a refinery acid waste stream), a smelter, and the facility sanitary wastewater. Existing treatment includes lime addition, polymer addition, flocculation, and settling. The pilot-scale treatment schemes investigated at this facility consisted of polishing technologies for improved metals removal.

A characterization of the untreated mine/mill/smelter/refinery wastewater during the period of the treatability study is presented in Table VIII-39 and a summary of the treated (existing) wastewater characteristics is given in Table VIII-40. The influent wastewater data was taken from analyses of daily composite samples (each daily non-flow proportional composite was composed of periodic grab samples taken over a three- to eight hour period). Treated effluent quality is the average of individually analyzed grab samples taken periodically each day over a two- to ten-hour period. Treated effluent samples taken from the facilities treatment plant represent the influent to the pilot plant system.

Sixteen treatability runs were completed during the test period (Table VIII-41). Test runs 01 through 05 used dual-media filtration at varying flow rates. Test run 06 used batch lime addition and one-hour settling. Test runs 07 through 09 included batch lime addition and flocculation followed by dual-media filtration at varying flow rates. Tests 10 through 13 represent one dual- media filter run at a fixed flow rate for an extended time (samples were taken after five minutes, six hours, 12 hours, and 18 hours). Runs 14, 15 and 16 were batch lime treated, followed by dual-media filtration and varying dosages of hydrogen peroxide for cyanide removal.

The following observations were made:

1. Dual-media filtration (tests 1-5) consistently achieved total suspended solids concentrations of less than or equal to 4 mg/l; total copper concentrations less than or equal to 0.11 mg/l; total lead concentration less than or equal to 0.018 mg/l; and total iron concentrations less than or equal to 0.09 mg/l. Zinc was less efficiently removed (influent mean = 0.309 mg Zn/l).

2. Lime addition with flocculation and secondary settling (one test run, test 6) achieved a total suspended solids concentration of 8 mg/l; total copper concentration of 0.25 mg/l; total lead concentration of 0.04 mg/l; and total zinc concentration of 0.17 mg/l.

3. Dual-media filtration with lime addition to a pH of approximately 9.0 (tests 7 through 9) achieved total suspended solids concentrations less than or equal to 1 mg/l; total copper concentrations less than or equal to 0.005 mg/l; and total zinc concentrations less than or equal to 0.043 mg/l.

4. During the 18 hour filter run (tests 10 through 13) no solid breakthrough occurred and metals concentration were relatively steady.

5. Lime addition to pH 10 and filtration (test 16) showed no improvement over the pH 9 tests.

No significant changes were observed in any of the remaining pollutants measured. A summary of pH, TSS, Cu, Pb, and Zn treatability study effluent concentrations is displayed in Table VIII-41.

Copper Mine/Mill/Smelter/Refinery 2121. The wastewater treatment system at this facility receives water from a smelter, a refinery, and a sanitary sewer. The combined flow is discharged to a tailing pond. Decant from the tailing pond passes through a series of five stilling ponds before final discharge. Table VIII-42 characterizes the facility's wastewater discharge during the pilot-scale treatability study conducted by Frontier Technical Associates. The quality was very good and represents the influent to the treatability study.

The study conducted included three test runs. Tests 01 and 02 were dual media filtration tests at hydraulic loadings of 6.5 gpm/ft^2 and 9.3 gpm/ft^2 respectively. Test 0.3 combined lime addition to a pH of 8.8 with dual media filtration at a hydraulic loading of 9.1 gpm/ft^2.

The results of sample analyses indicate no significant change in the already low concentrations of most pollutants. TSS levels dropped from an average of 4.1 mg/l to an average of 1.3 mg/l. In test 03, the pH dropped to 7.7 through the filter column, while causing partial clogging of the filter. Lime was visible in the filter effluent. This phenomenon presumably is due to the low solubility of lime in the facility wastewater which is high in sodium and calcium chloride salts. A summary of treatability study effluent sampling data is presented in Table VIII-43.

HISTORICAL DATA SUMMARY

This subsection presents long-term monitoring data gathered from individual facilities in several subcategories. Facilities considered here include those for which long-term data are available and which are regularly achieving or surpassing BPT limitations by optimizing their existing treatment systems.

Iron Ore Subcategory

Mine/Mill 1108 is located in the Marquette Iron Range in northern Michigan. The ore body consists primarily of hematite and magnetite and is mined by open-pit methods. Approximately 8,800,000 metric tons (9,700,000 short tons) of ore are mined yearly. The concentration plant produces approximately 2,800,000 metric tons (3,100,000 short tons) of iron ore pellets annually. Wastewater is presettled prior to treatment with alum and a long chain polymer to promote flocculation and improve settling characteristics. The treated water is polished in additional small settling ponds prior to discharge.

Table VIII-44 is a summary of industry supplied monitoring data for the period January 1974 through April 1977. As indicated, pH is well controlled and always in the range of 6 to 9. Alum and a polymer have been used on a continuous basis since 1975 to improve TSS removal at this facility. Since that time, TSS control has been excellent and has exceeded 30 mg/l only three times on a daily maximum basis. On a monthly average basis, this facility has exceeded 20 mg TSS/l only once since 1975. Dissolved iron concentration averages approximately 0.36 mg/l. Since 1975, dissolved iron concentration has not exceeded 2.0 mg/l on a daily basis or 1.0 mg/l on a monthly basis.

Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum Ore Subcategory

Copper/Silver Mine/Mill/Smelter/Refinery 2121

This facility is located in northern Michigan, with copper and silver ore extracted by underground methods. Mine production in 1976 was approximately 3,281,000 metric tons (3,617,000 short tons) of ore with 125,000 metric tons (138,000 short tons) of copper concentrate, and 185 metric tons (204 short tons) of silver concentrate. The primary mineral form is chalcocite. Concentration is accomplished by the froth flotation process. The smelter and refinery contribute wastewater to the combined treatment system. Wastewater also originates from power generation, sewage treatment, and collection of storm runoff. Wastewater from the above sources is combined in a tailing pond and decanted to a series of small settling basins before final discharge. The alkaline pH of the treatment system is maintained by the alkaline nature of the discharge from the mill as well as by the addition of lime to the slimes fraction of the tailings. The limed slimes are combined with all other wastewater sources in a mixing basin and then pumped into the tailing pond. The mine water contribution to the total discharge ranges from 0 to 4,500 m³/day (0 to 1.2 million gal/day), and this mine waste stream is released into the tailing pond on a seasonal basis. The total pond discharge volume averages approximately 79,000 m³/day (approximately 21 million gal/day).

Discharge monitoring data supplied by the company for a 58-month period between March 1975 and December 1979 are presented in Table VIII-45. This summary presents data derived from monthly averages for all parameters. The data presented for pH and TSS represent almost continuous daily monitoring throughout the reporting period. For these two parameters, the values shown are based on approximately 1,500 measurements. These data indicate that, for the parameters monitored, effluent performance is consistently far below BPT effluent standards, even when the maximum values reported are considered.

Wastewater treatment practices at Mine/Mill/Smelter/Refinery 2121 which are employed to attain its high quality effluent are:

1. Supplemental lime addition for improved coagulation, metals removal, and pH control

2. Use of a multiple pond system for improved settling conditions and system control

3. Sufficient pond volume to provide adequate retention time for sedimentation of suspended particulates and metals

4. Provision of a tailing pond design resulting in relatively efficient and undisturbed sedimentation conditions

5. Use of a decant configuration which effectively controls pond levels without disturbing settled solids in the vicinity of the decant towers

6. Mixing all waste streams prior to entry into the tailing pond system to reduce the possibility of thermal stratification and pH fluctuations.

Copper Mine/Mill 2120

This mine/mill facility is located in southwest Montana. The ore body consists primarily of chalcocite and enargite, mined only by open-pit methods at present. Underground mines at this facility are inactive, but mine water is continuously pumped. The mill employs the froth flotation process to produce copper concentrate, while cement copper is produced by dump leaching of low grade ore. In 1976, ore production was 15,419,000 metric tons (17,000,000 short tons), and 327,000 metric tons (360,000 short tons) of copper concentrate were produced. Approximately 16,000 metric tons (17,600 short tons) of cement copper are produced annually.

Schematics of the wastewater treatment system employed at Mine/Mill 2120 are presented in Figures VIII-8 and VIII-9. .Figure VIII-8 portrays the system configuration as it existed during the period (i.e., September 1975 through June 1977) when the data presented in Table VIII-46 were collected.

Wastewater streams routed to the tailing pond system for treatment include underground mine water, excess leach circuit solution, and mill tailings. The mine water is acidic because sulfuric acid is added to prevent iron-deposit fouling of pipes and pumps used for mine dewatering. The acidic leach circuit waste stream results as a 3 percent bleed from a 190,000 m³ (50 million gallons) of solution recycled through the leach circuit daily. Reportedly, this bleed is used because seepage into the dump leach system necessitates discharge of excess water. Additional lime is added to the mill tailings to neutralize the acidity of the mine water and leach solution. These three waste streams are thoroughly mixed prior to combined discharge into the tailing pond. Prior to 1977 the tailing pond decant was largely recycled to the mill for use as process water, but tailing pond overflow was discharged when effluent quality permitted. When tailing pond decant was discharged, the average .daily discharge volume was 11,000 m³ (3 million gallons).

When tailing pond overflow quality did not permit discharge, wastewater was reintroduced into the mill circuit and subsequently mixed with open-pit mine water for additional treatment in a second treatment system (i.e., the "barrel pond" system). This treatment system consists of a three celled settling pond where the influent wastewater is limed and polymer is added to enhance flocculation and settling. A relatively high pH is maintained through this treatment system, but a final pH adjustment is made when necessary by addition of sulfuric acid. Average discharge volume from this treatment system is approximately 25,000 m³ (6.5 million gallons) per day.

Figure VIII-9 shows modifications which have been made to the treatment system at Mine/Mill 2120 since June 1977. Although direct discharge of tailing pond decant has not occurred during the past two years, discharge could occur if excess water conditions warrant. Notable among the changes made to the treatment system is the addition of a pond for secondary settling of tailing pond decant before recycle. In addition, open pit mine drainage has been directed to the tailing thickeners to avoid surge and overflow. However, mine/mill personnel report that overflow from the surge pond still occurs intermittently and is still combined with treated effluent from the barrel pond system for final discharge. As will be discussed, this latter practice has an adverse impact on the quality of the combined discharge stream.

Tailing pond effluent monitoring data supplied by industry are presented in Table VIII-46. These data have been summarized for the period September 1975 to June 1977 on the basis of both daily averages and averages of monthly means. It is noted that the pH of the tailing pond effluent falls outside of the BPT limits much of the time, but a high pH level is reportedly maintained to improve pH values downstream of the discharge point with the consent of the state.

Practices which have been identified as essential to the attainment of consistent and reliable treatment in the tailing-pond system are:

1. Maintenance of pH slightly in excess of 9.

2. Maintenance of an earthern dike (baffle) within the tailing pond to prevent short circuiting and reduce wind induced turbulence.

3. Discontinuation of tailing pond discharge during upset conditions.

Effluent monitoring data describing the quality of effluent discharged from the second treatment system (i.e., the barrel pond system) are presented in Table VIII-47, a summary for the period January 1975 to September 1977.

It is important to note that the data presented in Table VIII-47 do not accurately reflect the capabilities of the barrel pond treatment system. The reason for this, as indicated in Figure VIII-9, is that untreated wastewater is often combined with treated effluent prior to final discharge. (The effluent monitoring station is located downstream of the point where these waste streams are combined.) This practice adversely impacts the quality of final discharge and is considered to be primarily responsible for the BPT violations. (The exception is pH, which reportedly is purposely maintained at a high level to improve acid conditions in the receiving stream.) Industry personnel report that actions are presently being taken to eliminate the necessity for this practice.

Lead/Zinc/Copper Mine/Mill 3105

This underground mine is located in Missouri. Galena, sphalerite, and chalcopyrite (lead, zinc, and copper minerals) are the primary minerals recovered. Ore production began in 1973, and reported mine production was 1,032,000 metric tons (1,137,700 short tons) in 1976.

Mining and milling wastewater streams are treated separately. The mill operates in a closed loop system; tailings are treated in a tailing pond, and the pond decant is recycled back to the mill. Some mine water is used as makeup water in the mill flotation process. Excess mine water, averaging 8,300 cubic meters (2.1 million gallons) per day is treated by sedimentation in a 11.7 hectare (29 acre) settling pond.

Effluent monitoring data for the mine water treatment system at Mine 3105 are presented in Table VIII-48. This data summary is based on NPDES monitoring reports submitted for this facility for the period January 1974 through January 1978.

The mine is an example of low solubilization of heavy metals due to the mineralization of the ore body. More specifically, the ore body is low in pyritic minerals and exists in a dolomitic host rock. Mines exhibiting low solubilization potential are characterized by mine waters of near neutral to slightly alkaline pH.

Mine water treatment at Facility 3105 illustrates that simple sedimentation at mines exhibiting low solubilization potential may be sufficient to achieve water quality superior to BPT limitations, by effective removal of suspended solids and associated particulate metals (see Table VIII-48).

Lead/Zinc/Silver Mine 3130

This facility is located in Utah and produces ore with economic mineral values of sphalerite, galena, and tetrahedrite in a quartz and calcite matrix. Production at this facility is confidential. No discharge occurs from the associated mill by virtue of process wastewater impoundment and solar evaporation. Mine water pumped from this operation averages 32,700 m^3 (8.64 million gallons) per day. The mine water treatment system consists of lime and coagulant addition, followed by multiple-pond sedimentation. Backfilling the mine with the sand tailing fraction from the milling circuit is practiced. Since the associated mill utilizes sodium cyanide in the flotation process, the sand tailings used for backfilling contribute cyanide to the mine water discharge. This cyanide is not effectively removed by the treatment system. The problem of cyanide in mine water resulting from operations which practice cut and fill techniques has been discussed in Section VI.

Tables VIII-49 and VIII-50 summarize data on raw and treated waste streams by industry for the period June 1977 to October 1977. A new treatment system was recently brought online, so a great deal of data are not available. Examination of raw waste data indicates that the mine water contains high concentrations of metals. As shown in Table VIII-50, the careful control of pH, use of a settling aid (i.e., polymer), and use of a multiple pond settling system have resulted in effective removal of metals and suspended solids during the period reported.

Zinc/Copper Mine/Mill 3101

This mine/mill facility is located in Maine. Ore mined from underground contains sphalerite and chalcopyrite (also minor amounts of galena). Zinc and copper concentrates are produced in the mill by the flotation process. In 1973, mine production totaled 209,000 metric tons (231,000 short tons) of ore. Zinc and copper concentrate production from the mill totaled 25,600 metric tons (28,200 short tons). Operations were suspended at this facility in October 1977 due to the depressed copper and zinc markets.

For the most part, mine water was used in the mill flotation circuit. Mill tailings and any mine water not used in the mill were discharged to a primary tailing pond having an area of about 20.2 hectares (50 acres). Decant from this pond flowed into an auxiliary pond, approximately 3.2 hectares (8 acres) in area, to a pump pond approximately 0.81 hectare (2 acres) in area, and was discharged. The pH of the final discharge was continually monitored and adjustments were made to optimize removal of metals (especially zinc, iron, and manganese), and to maintain the pH within limits specified by state and federal permits.

Tailings discharged from the mill flotation circuits had a pH in the range of 9.9 to 11.7. This was largely due to the use of lime as a depressant in the zinc flotation circuit. Additional lime was occassionally added to the tailings. On weekends, when the mill was not operating, lime was added to the excess mine water, which was discharged to the tailing pond system. During the coldest months of the year (January, February, and March), problems were encountered with maintenance of the final effluent pH within the required 6 to 9 range. During this period, the 30-day average pH is often as high as 10.7. For this reason, no lime, other than that used in the mill flotation circuits, was added to either the tailings or the excess mine water during these months.

Because available mine water did not provide the total volume of water required in the mill, part of the treatment system effluent was recycled. Approximately 56 percent of the mill feed water was obtained in this manner.

Other wastewater control technologies included the segregation of spills from the copper and zinc flotation circuits and control of surface drainage with ditches and surface grading.

Effluent data submitted by the company for the period of January 1974 to August 1977 are summarized in Tables VIII-51 and VIII-52. These data consistently demonstrate achievement of effluent quality superior to that specified by BPT guidelines, with the exception of pH. Severe pH excursions occur in the winter months, and this phenomenon is not clearly understood.

Comparison of Tables VIII-51 and VIII-52 illustrates the improvements in water quality as it passed through a multiple pond system. Note the reductions in the percentage of time the quality is out of compliance at the tailing pond decant compared to the final discharge. The merits of the multiple pond treatment system are further substantiated by the reduced average concentrations and variability illustrated by the data describing the secondary pond effluent.

Wastewater treatment practices at Mine/Mill 3101 considered to be important to consistent and reliable attainment of a high quality effluent are:

1. Maximum utilization of mine water in the mill flotation circuits, thus minimizing wastewater flows to be treated

2. Supplemental lime addition (after flotation) for optimum metals precipitation

3. Use of the multiple pond system for improved sedimentation conditions and improved system control

4. Provision of ponds of sufficient size (volume) to provide adequate sedimentation conditions and long-term storage capacity

5. Segregation and recycle of spills and washdown water in the mill

6. Combined treatment of mine and mill wastewater streams for improved metals removal

Lead/Zinc Mine/Mill 3102

This facility is located in Missouri and produces the largest output of lead concentrate and the second largest output of zinc concentrate in the United States. Approximately 1,482,000 metric tons (1,635,000 short tons) of ore are mined annually at this facility, with sphalerite, galena, and chalcopyrite as the primary ore minerals. In 1975, 228,600 metric tons (252,100 short tons) of lead concentrate and 41,600 metric tons (45,900 short tons) of zinc concentrate were produced at the flotation mill.

Wastewater treatment consists of alkaline sedimentation of combined mining and milling wastewater streams in a multiple pond system. The exclusive use of mine water as the process and potable water supply for the mill reduces the hydraulic loading percent. Since the mine produces more water than the mill can use, the excess mine water is discharged to the tailing pond for treatment.

The mill slime tailings are discharged to the main tailing pond after separation (by hydrocyclones) of the sand fraction for dam building. The tailing pond now occupies about 32.4 hectares (80 acres) and will occupy 162 hectares (400 acres) when completed. The decant from this pond flows into a small stilling pool, then through a series of shallow meanders, to a polishing pond of approximately 6.1 hectares (15 acres), and is subsequently discharged.

A summary of effluent monitoring data for the period of December 1973 through September 1974 is presented in Table VIII-53. These data indicate that all parameters analyzed are several orders of magnitude lower than BPT limitations.

Zinc/Lead/Copper Mine/Mill 3103

This facility is located in Missouri and has an underground mine. The minerals of principal value are galena, sphalerite, and chalcopyrite. Zinc, lead, and copper concentrates are produced by the flotation process in the mill. In 1976, mine production totaled 972,300 metric tons (1,072,400 short tons), while 92,400 metric tons (102,000 short tons) of lead concentrate, and 9,800 metric tons (10,800 short tons) of copper concentrate were produced at the mill.

Mine and mill wastewater streams are combined for treatment at this facility, as indicated in Figure VIII-10. Wastewater treatment consists of alkaline sedimentation in a multiple pond settling system.

Wastewater discharge volume is minimized by the extensive use of mine water and tailing pond recycle as flotation makeup water in the mill. Combined influent flow to treatment averages 10,900 m^3 (2.88 million gallons) per day, of which 5,450 m^3 (1.44 million gallons) per day are recycled when the mill is operational. Throughout most of the year, the lime added to the flotation circuit is considered (by plant personnel) to be sufficient to produce a wastewater pH high enough for effective heavy metals removal. However, during cold winter months, as much as 0.9 to 1.8 metric tons (1 to 2 short tons) of additional lime are added daily to the mill tailings to suppress rising heavy metal concentrations, especially zinc, in the final effluent.

Summaries of effluent monitoring data are presented in Tables VIII-54 and VIII-55. These summaries are based solely on analytical data provided by industry. These data reveal several important points relative to the treatment system performance and capabilities at Mine/Mill 3103:

1. The effluent from the secondary settling pond (Table VIII-55) was far below BPT limitations (monthly mean), sometimes by an order of magnitude for all control parameters (pH, TSS, lead, zinc, copper, cadmium, mercury, and cyanide) for the period February 1974 through November 1977.

2. The tailing pond effluent was in compliance with BPT limitations (monthly mean) for pH, TSS, and copper 100 percent of the time during the same 46-month period. Only two of the 40 observations, or 5 percent, were above the limitations for both lead and zinc (not necessarily in the same sample).

3. Both the mean and the standard deviation (variability) of all metals data were significantly less in the effluent from the second settling pond than in the effluent from the tailing pond.

The factors contributing to the effluent quality attained at Mine/Mill 3103 are:

1. The multiple pond treatment system;

2. Extensive use of mine water and tailing pond recycle in the mill;

3. Combined treatment of excess mine water, concentrate thickener overflow, and mill slime tailings; and

4. Supplemental lime addition for metals removal when necessary.

Lead/Zinc Mine/Mill 3104

This facility is located in northern New York State. Ore mined from an underground mine contains sphalerite and galena. Zinc and lead concentrates are produced by the flotation process in the mill. Mine production was 1,009,100 metric tons (1,110,000 short tons) of ore in 1973, while the mill produced 113,100 metric tons (124,400 short tons) of lead and zinc concentrates that year.

Approximately 6,820 m³ (1.8 million gallons) of wastewater per day are treated by alkaline sedimentation at this facility. The tailing pond has a total impoundment area of 32 hectares (80 acres). This area is divided into three ponding areas, which are 4 hectares (10 acres), 15 hectares (37 acres), and 13 hectares (33 acres) in area, respectively. Recent modifications at this operation include partial recycle of treated effluent during summer months and the use of all mine water as mill feed, thus eliminating mine water discharge.

Table VIII-56 summarizes tailing pond effluent data for this treatment system for the period January 1974 to December 1977. An examination of these data indicates that total metal values are well within the BPT limits even when the maximum values reported are considered. The TSS concentrations average approximately 7 mg/l, with a maximum reported monthly value of 16 mg/l.

The factors contributing to the effluent quality attained at Mine/Mill 3104 are:

1. Maximum utilization of mine water in the mill, which reduces the volume of wastewater requiring treatment, and

2. A tailing pond configuration designed to minimize short circuiting, with provision of adequate impoundment volume to promote effective sedimentation.

These practices have eliminated the discharge of mine water and, thus, reduced the total volume of wastewater to be treated and discharged. Although the pH attained in the tailing pond is not considered to be optimum for metals removal, the alkalinity of the mill tailings is sufficient to reduce dissolved metals to levels consistently better than BPT limitations without supplemental lime addition or extensive pH control.

Zinc Mill 3110

This flotation mill is located in central New York and beneficiates an ore which contains sphalerite and pyrite as the major minerals in a dolomitic marble. Minor constituents of lead, cadmium, copper, and mercury are also present. In 1976, the mill recovered 118,000 metric tons (13,000 short tons) of zinc concentrate from 93,900 metric tons (103,300 short tons) of ore. An average of 830 m³ (220,000 gallons) per day of wastewater is pumped from the mine to the mill water supply reservoir for use as mill makeup water. The mill water supply is augmented by other fresh water sources as required. The mill discharges 990 tailing deposition area. Mill water flows over and percolates through the deposited tailings and is collected in a 3.2-hectare (8 acre) settling pond. Decant from this pond flows by gravity into a 1.2 hectare (3 acres) pond, followed by a third 8.1hectare (20 acre) settling pond. (The third pond was not constructed by the operators, but exists due to a beaver dam.) Due to the influence of surface drainage, daily discharge volume from the treatment pond system averages 2,650 cubic meters (650,000 gallons) per day.

Table VIII-57 presents a summary of company monitoring data for the period January 1974 to April 1977. These data represent grab samples collected once monthly for 40 months. All parameters analyzed were well below BPT limitations throughout the monitoring period.

Molybdenum Mine 6103

This operation, located in Colorado, is a recently opened underground mine yielding molybdenum ore at the rate of approximately 2,200,000 metric tons (2,425,000 short tons) per year. A discharge of 9,100 m^3 (2.4 million gallons) per day is treated by spray cooling, and suspended solids are removed in a multiple pond system with the aid of flocculants prior to discharge. The mill which recovers molybdenite by flotation, is located some distance from the mine and is connected to the mine by a long haulage tunnel. Extensive recycle is practiced, and there is no wastewater discharge at the mill site.

Table VIII-58 summarizes the limited data provided by the company for the period July 1976 to June 1977. In general, TSS concentrations are well below 20 mg/l. Effluent metal values are reduced substantially below BPT limitations.

Molybdenum Mill 6101

This facility, which uses the flotation process to concentrate molybdenum ore, is located in mountainous terrain in New Mexico. Ore is obtained from a large open-pit mine, with production at 5,700,000 metric tons (6,300,000 short tons) per year. The flotation mill produces an alkaline tailings discharge which flows approximately 16.1 kilometers (10 miles) to the tailings disposal area, where sedimentation in primary and secondary settling ponds takes place. The average discharge volume from this treatment system is 11,000 cubic meters (4.6 million gallons) per day.

Table VIII-59 summarizes effluent monitoring data for the period January 1975 through December 1976. Values reported are substantially below BPT limitations. Recently, this operation used hydrogen peroxide addition to the tailing pond decant stream during cold and inclement weather for the control of cyanide discharges on an experimental basis. The effect of this treatment, not reflected in the data presented in Table VIII-59, is, according to mine personnel, the reduction of cyanide concentrations from approximately 0.05 mg/l to less than 0.02 mg/l.

Molybdenum Mine/Mill 6102

This facility is located in Colorado and employs both open pit and underground mining methods. Approximately 14,000,000 metric tons (15,400,000 short tons) of ore containing molybdenum, tungsten, and tin are processed each year. The ore is beneficiated at the site by a combination of flotation, gravity separation, and magnetic separation methods to produce concentrates of molybdenum, tungsten, and tin.

A daily average of 3,800 cubic meters (1 million gallons) of mine water is pumped from the underground workings to the mill tailing ponds. Three tailing ponds receive the mill tailings discharge, and most of the clarified effluent is recycled to the mill. The system of tailing ponds, impoundment, and extensive recycle has been used to achieve zero discharge throughout most of the year. Heavy snowmelts flowing to the treatment system have necessitated a discharge during the spring of most years. Extensive runoff diversion works have been installed to reduce spring discharge volume. The treatment system includes ion exchange for molybdenum removal, electrocoagulation flotation removal of heavy metals, alkaline chlorination for the destruction of cyanide, and mixed media filtration. A continuous bleed through this treatment system will replace the previous seasonal discharge to limit the required capacity and, thus, the capital costs.

Full scale operation of the treatment system described above was initiated during July 1978. This treatment system is designed to treat 7.6 cubic meters (2,000 gallons) per minute; however, at the date of sampling, the system had been operated at only 3.8 cubic meters (1,000 gallons) per minute. The following discussion of this treatment system reflects its performance during the first four months of its operation.

The treatment facility houses all the aforementioned unit processes and is located below the series of tailing ponds. Feed for the system is a bleed stream from a final settling pond whose characteristics are presented in Table VIII-60.

The wastewater is treated first in an ion exchange unit (pulsed bed, counter-flow type) to remove molybdenum. This ion exchange unit uses a weak-base amine-type anion exchange resin for optimum molybdenum adsorption. The influent is acidified to approximately pH 3.5, since molybdenum adsorption is reported to be most efficient at a pH in the range of 3.0 to 4.0 (Reference 67). Initial results indicate that an influent molybdenum concentration of 5.6 mg/l is reduced to 1.3 mg/l in the ion exchange effluent. Molybdenum recovery from the eluant (backwash) has not been practiced to date. When the system is optimized, molybdenum recovery is planned. However, several problems with the columns (most notably, excessive pressure at flow exceeding 3.8 cubic meters, or 1,000 gallons, per minute) have impeded the assessment of the actual treatment capability of this unit process.

The ion exchange effluent is treated by electrocoagulation flotation for removal of heavy metals. This process involves the formation of a metal hydroxide precipitate (by addition of lime), which is then conditioned in an electrocoagulation chamber via contact with hydrogen and oxygen gases, generated by electrolysis. The resulting slurry is mixed with a polymer flocculant and floated in an electroflotation basin by small bubbles of oxygen and hydrogen. The floated material is skimmed off and discarded. To date, the effluent from this process has been monitored only for TSS, iron (total), and cyanide. The extent to which these parameters have been removed by the electrocoagulation flotation process is indicated by the following:

Parameter	Concentration (mg/l) Influent to Electrocoagulation	Effluent from Electrocoagulation
TSS	127	65
Fe (Total)	1.8	0.5
Cyanide	0.09	0.04

Total system effluent monitoring data indicate that effective removal of zinc and manganese is also attained (refer to Table VIII-60). Efficient dewatering and handling of the sludge which results from this process have not been optimized and this problem has not been resolved.

Effluent from the electrocoagulation flotation process is treated by alkaline chlorination for destruction of cyanide and then polished by mixed-media filtration prior to final discharge. The sodium hypochlorite used for the alkaline chlorination is generated on-site by the electrolysis of sodium chloride. The hypochlorite is injected into the waste stream prior to the filtration step. The first four months of data indicate that influent cyanide levels (clear pond bleed) range from less than 0.01 to 0.20 mg/l while the treatment-system effluent concentrations of cyanide range from less than 0.01 to 0.04 mg/l. After the treatment plant effluent passes through a final retention pond (residence time of approximately 2 hours), the cyanide levels are consistently below 0.01 mg/l. The retention pond was added to the system to ensure adequate contact time for the oxidation reaction to occur. Since the system is still in the process of optimization, it is expected that dosage levels for the hypochlorite will be optimized, and that possible problems with high levels of residual chlorine will be eliminated.

Mixed-media filtration was incorporated into the treatment scheme to provide effluent polishing for optimum removal of suspended solids and metals.

In spite of difficulties which have been encountered the overall performance of the treatment system has been good (refer to Table VIII-61). Plant personnel report that the effectiveness of the treatment system at this time has generally exceeded their expectations based on pilot plant experience.

Aluminum Ore Subcategory

Open-pit Mine 5102 is located in Arkansas and extracts bauxite for metallurgical production of aluminum. Approximately 900,000 metric tons (approximately 1,000,000 short tons) of ore are mined annually at this site. A bauxite refinery which produces alumina (Al_2O_3) in a variety of forms and which recovers gallium as a byproduct is located on site, but no wastewater from the refining operation is contributed to the mine water treatment system. Bauxite mining at this operation occurs over a large expanse of land, and several mines may be worked at one time. Because of the long distance between mine sites, several mine water treatment plants have been constructed. There are three mine water discharge points averaging 10,900 cubic meters (2.8 million gallons) per day, 14,100 cubic meters (3.7 million gallons) per day, and 7,000 cubic meters (1.9 million gallons) per day, respectively. The associated wastewater treatment systems consist of lime addition and settling. Monitoring data for each of the discharges are presented in Tables VIII-62 through VIII-64. Each of the three discharges consistently meets BPT daily average and monthly maximum total suspended solids concentrations.

Mine 5101 is an open pit mine located adjacent to Mine 5102 in Arkansas. Bauxite is mined at this facility for the production of metallurgical aluminum. Approximately 900,000 metric tons (1,000,000 short tons) of ore are mined yearly. The ore is hauled directly to the refinery. There are presently three active discharge streams with separate treatment systems employing similar treatment technologies. Lime addition and settling are used to treat the acid mine drainage of Mine 5101. Portable, semi-portable, and stationary treatment systems are all currently being used at this mine. Monitoring data for each of the discharges are presented in Tables VIII-65 through VIII-67. Each of the discharges consistently met BPT limitations for total suspended solids and aluminum during the monitoring period.

Tungsten Ore Subcategory

Tungsten Mine/Mill 6104

This operation is located in California in mountainous terrain at elevations of 2,400 to 3,600 meters (8,000 to 11,000 feet). A complex tungsten, molybdenum, and copper ore is mined at the rate of 640,000 metric tons (700,000 short tons) per year. A large volume of mine water, 38,000 m^3 (10 million gallons) per day, flows by gravity from the portal of this underground mine. Approximately 20 percent of this flow is used in the mill. The remainder is treated for suspended solids removal in a clarifier and discharged to a stream. The mill at this site uses several stages of flotation to yield concentrates of molybdenum and copper, and a tungsten concentrate which is further processed by leaching and solvent extraction to yield purified ammonium paratungstate. All mill effluent flows to a series of three tailing ponds which have no surface discharge.

Table VIII-68 summarizes treated mine water effluent monitoring data for the period from Jaunary 1976 through December 1976. As the data show, this effluent contains extremely low concentrations of most pollutants. The treatment system provides effective control of TSS and, consequently, of most metals present in the effluent. Molybdenum occasionally occurs at measurable concentrations.

Uranium Ore Subcategory

Uranium Mine 9408

This operation recovers uranium from a hard-rock, underground mine in Colorado. The principal uranium mineral found in the vein-type deposits is pitch blende, in association with carbonates and pyrite. The ore contains an average of 0.3 percent U_3O_8 and must be shipped approximately 200 kilometers (190 miles) to the associated mill. Therefore, it is crushed and sorted on-site to increase grade. The ore finally shipped contains an average of 0.6 percent U_3O_8.

Approximately 3,500 cubic meters (940,000 gallons) per day of mine water and a small volume of sanitary wastes are combined and directed to the wastewater treatment plant. Treatment consists of chlorination with sodium hypochlorite to disinfect the sanitary wastes, coagulation with an anionic polymer, barium chloride coprecipitation for radium removal, and settling. Settling takes place in a series of two concrete-lined basins and four ponds with a combined capacity of 4,700 m^3 (1,250,000 gallons). Settled solids are periodically removed and trucked to the associated mill for recovery of residual uranium and subsequent disposal in mill tailings.

A summary of company reported effluent monitoring data is presented for the period April 1975 through January 1977 in Table VI-69. All parameters are well below BPT limitations. The data demonstrate correlation between control of suspended solids and total Ra 226.

Concern has been expressed over the applicability and efficiency of this treatment system to mine water from the more common sandstone deposits. However, similar facilities treating mine water from sandstone deposits are achieving effective removal of radium 226 also.

Nickel Ore Subcategory

Nickel Mine/Mill 6106

This facility is located in Oregon and produces ferronickel directly by smelting 3,401,000 metric tons (3,746,500 short tons) per year of lateritic ore from an open-pit mine. Mine area runoff, ore and belt wash water, and smelter wastewater are combined and treated in a series of two settling ponds. A considerable volume of water is recycled to the smelter from the second of these ponds, and surface discharge from the third pond occurs intermittently, depending on seasonal rainfall.

Available monitoring data submitted by the company for the period of January 1976 through December 1980 are summarized in Table VIII-70. Because discharge from the ponds is intermittent, the data represent the quality of surface water in the final settling pond from which discharge occurs.

Figure VIII-11 is a plot of the concentrations of selected effluent constitutents versus time. These data illustrate the importance of seasonal meteorological conditions to many facilities. At this site, mine runoff during the rainy season (approximately November through April) significantly increases flow through the settling ponds, thus reducing residence time, adversely affecting secondary settling and increasing the concentration of TSS in the effluent.

Vanadium Ore Subcategory

Vanadium Mine 6107

Mine 6107 is an open pit vanadium mine located in Arkansas. Opened in 1966, this mine annually produces approximately 363,000 metric tons (400,000 short tons) from a non-radioactive vanadium ore.

Mine area runoff, waste pile runoff, and seepage from the waste pile are collected and treated in a common system. Mine area and waste pile runoff are diverted to the wastewater treatment plant. Seepage from the waste pile is collected in several small ponds and pumped to the treatment plant. At the treatment plant, lime is mixed with the wastewater to adjust the pH to within the range of 6.0 and 9.0. Wastewater from the treatment plant flows into a large settling pond which was formerly an active pit. Depending upon the water quality, the effluent from the settling pond is either discharged or recycled to the treatment plant.

A summary of discharge monitoring data from Mine 6107 between July 1978 and December 1980 is presented in Table VIII-71. Relatively low concentrations of suspended solids were consistently reported. The average total iron concentration was 0.65 mg/l. The pH ranged from 5.4 to 9.3 with only two excursions from the range of 6.0 to 9.0.

Titanium Ore Subcategory

Titanium Mine/Mill 9906

Mine/Mill 9906 is a titanium dredge mining and milling operation located in Florida and adjacent to titanium Mine/Mill 9907. Ilmenite ore from a placer deposit is dredged from a man-made pond. Humphrey spirals located on a floating barge behind the dredge are used to concentrate the heavy minerals in the ore. The lighter minerals are returned directly to the dredge pond. Electrostatic and magnetic separation methods are utilized to further concentrate the ilmenite.

Excess mine water, runoff, and mill wastewater from the caustic pond overflow are combined and treated in a common system. The first step in the treatment process consists of lowering the pH to approximately 4.0 with a strong acid to assist in coagulation of the organic material. The wastewater then flows through a series of settling ponds, after which the pH is adjusted upward to meet discharge limitations. The wastewater then flows through a series of small ponds before final discharge.

A summary of reported effluent monitoring data is presented in Table VIII-72. The average discharge rate was approximately 26,000 cubic meters (6.85 million gallons) per day. The average TSS concentration was less than 10 mg/l. The pH concentration ranged from 4.0 to 10.0 and averaged 7.0 with few excursions.

ADDITIONAL EPA TREATABILITY STUDIES

Copper Mill 2122

Tailings from bulk copper flotation circuits located in two copper mills at this facility are discharged to a 2,145-hectare (5,300-acre) tailing disposal area for treatment. Due to the design and mode of operation of this tailing disposal area, the effluent quality attained is often very poor. Wind disturbances, short-circuiting of the settling pond (the area actually covered by standing water is 101 hectares, or 250 acres and the depth of water is only a few centimeters over much of this area), and a floating-siphon effluent system that at times pulls solids off the bottom of the pond are all factors which frequently produce high total suspended solids-concentrations in the tailing pond effluent. For this reason, the unit processes investigated at this facility were flocculant (polymer) addition, flocculation, secondary settling, and filtration. Lime addition was also investigated to determine possible benefits derived in terms of metals removal. Experiments employing various combinations of these unit processes were designed primarily to evaluate improvements in treatment efficiency attainable by addition of polishing treatment to the existing tailing pond system.

The treatability study at Mill 2122 was performed during two different time segments. These time periods were 5 to 15 September 1978 and 8 to 19 January 1979. During the September phase of the study all of the unit processes identified above were investigated. The purpose of the final phase was to further investigate the capabilities of dual-media filtration for removal of TSS and metals from the tailing pond decant at this site in addition conducting cyanide destruction studies.

Company personnel at Mill 2122 have previously reported that cyanide concentrations in the tailing pond decant are high enough to cause problems only during the winter months (i.e., December to March). For this reason, the second treatability study at Mill 2122 was scheduled for early January which was expected to be an optimum time to conduct cyanide destruction studies. However, the concentration of cyanide in the decant remained very low (i.e., less than 0.05 mg/l) throughout the January study period. For this reason, it was decided to spike the tailing pond decant with cyanide prior to the cyanide destruction experiments. Two unit processes, alkaline chlorination and ozonation were investigated for cyanide destruction capabilities.

Initially, four species of cyanide (i.e., calcium cyanide, sodium cyanide, ferrocyanide, and ferricyanide) were used for spiking, independently of one another, to investigate the impact of the chemical form of cyanide on the destruction technology capabilities. However, experiments with ferricyanide and ferrocyanide were discontinued after quality control results indicated almost no analytical recovery of cyanide from control samples spiked with these species and analyzed by the EPA approved Belack distillation method. All samples collected for cyanide analysis were analyzed within 24-hours by a local commercial laboratory.

Influent to the pilot plant was taken from the tailing pond effluent line. This line is used for recycle as well as for discharge. The character of the tailing pond effluent during the periods of study is presented in Tables VIII-73 and VIII-74. As can be seen from these tables, the concentrations of total suspended solids and total metals in the recycle water were highly variable during the period of the study. The consistently low concentrations of dissolved metals observed indicate that metals present in the tailing-pond discharge (i.e., recycle water) were contained in the suspended solids. This is further evidenced by the high correlation (r = 0.99) between total copper and TSS concentrations in the wastewater (see Figure VIII-12). This relationship suggests that any polishing treatment which effectively removes the suspended solids will also effectively remove the metals.

Results of the pilot-scale treatability studies are presented in summary form in Tables VIII-75, VIII-76, VIII-77, and VIII-78. A review of the results presented in Table VIII-75 indicates that secondary settling at a theoretical retention time of 10.4 hours was sufficient to produce effluent total metal concentrations well below BPT limitations. In a full scale system, even longer times (24 to 72 hours) would be recommended to reliably achieve this limit. A larger pond would also provide protection against surge loads and short-circuiting.

Polymer and lime addition, followed by flocculation prior to settling (2.8 hour retention time), produced effluent suspended-solids concentrations comparable to those achieved by secondary settling with a longer retention time (10.4 hours). The observed improvement in efficiency of removal of TSS at shorter retention time is attributed to the addition of polymer. This is further evidenced by the fact that treatment schemes employing lime addition and settling resulted in much higher suspended solids levels in the effluent when a polymer was not employed.

Experiments employing lime addition were conducted to investigate its effect on dissolved metal precipitation and suspended solids settleability. However, because dissolved metal concentrations in the tailing pond recycle water were already very low (i.e., less than 0.04 mg/l), this treatment provided little or no benefit.

Three dual-media, downflow, pressure filters consisting of different filter-media sizes and depths, were evaluated over a a range of hydraulic loadings of 117 to 880 m³/m²/day (2 to 15 gpm/ft²). All three filters employed consistently produced filtrates with suspended solids concentrations of less than 10 mg/l throughout the range of 30 to 50 mg/l. On two occasions, however, filter performance was adversely impacted by shock loads. At these times, the suspended solids concentrations of the tailing pond recycle water being treated ranged upwards to several percent solids, and the filtrate concentrations attained were 13 and 30 mg/l.

Because dual media filtration at hydraulic loadings of 293 to 880 m³/m²/day (5 to 15 gpm/ft²) demonstrated consistently good removal of suspended solids during eight-hour runs, it was desired to investigate filter performance at a high hydraulic time. The results of this experiment are presented in Table VIII-75. As indicated, the TSS concentration of the tailing pond decant averaged 33 mg/l during this experiment. Total suspended solids concentrations of 7 to 12 mg/l were attained in the filter effluent during the first four hours of the run. However, solids breakthrough began to occur between the 4th and 7th hours of the run. Therefore, at a hydraulic loading of 13 gpm/ft² the frequency of backwash required appears to be much greater than the frequency required for a loading of 10 gpm/ft² or less.

Subsequent filtration experiments planned during January were long runs at 5 and 10 gpm/ft² to determine the frequency of backwashing required. An experiment at a loading of 4 gpm/ft² was initiated, but was terminated after one hour because solids breakthrough occurred almost immediately due to extremely high concentration of TSS (i.e., 1,200 mg/l) in the tailing pond decant being filtered. The use of dual-media filtration for effluent polishing is generally effective when the influent TSS concentrations are no greater than 35 to 50 mg/l. However, at the very high TSS concentrations which frequently occurred at Mill 2122 filtration is not feasible. Because high concentrations of TSS persisted in the tailing pond decant during the remainder of the final study period, no further filtration experiments were attempted.

To investigate alkaline chlorination a series of bucket tests were conducted to maximize the number of dosages, pH values and contact times which could be employed over a short period of time. As previously mentioned, meaningful results were not obtained from initial experiments in which ferricyanide or ferrocyanide were used as spikes due to the lack of quantitative analytical recovery of these cyanide species from untreated spike samples. For this reason, experiments with these cyanide species were discontinued.

Results of bucket tests in which sodium cyanide was used to spike the wastewater are summarized in Table VIII-70. These data indicate the most efficient destruction of cyanide occured at the highest hypochlorite dosages employed, i.e., 20 and 50 mg/l. At these dosages, significant differences between the various pH levels and contact times employed were not evident. At the lower hypochlorite dosages, 5 and 10 mg/l, good destruction of cyanide appeared to be achieved at pH 9. However, these data must also be viewed with caution as a quality control program conducted in conjunction with the sodium cyanide spike experiments indicated erratic and unreliable analytical recoveries (see Section V).

During the alkaline chlorination study the destructability of total phenol (4AAP) present in the tailing pond decant was observed. The data presented in Table VIII-70 indicate that effective destruction of total phenol (4AAP) occurred only at the highest hypochlorite dosage, 50 mg/l.

The literature indicates that oxidation of phenolic compounds with chlorine species may produce highly toxic chlorophenols. Although the production of these compounds was not investigated during this study, their potential production should be evaluated if full scale alkaline chlorination of a phenol containing waste stream is seriously considered.

Experiments to evaluate the destruction of cyanide by ozonation were conducted using a continuous flow pilot scale treatment system. Variables evaluated during the ozonation experiments included the weight ratio of ozone to cyanide maintained to the contact chamber, pH, and contact time. The results of these experiments are presented in Table VIII-78. These results indicated that destruction of cyanide occurred to varying degrees in the contact chamber. At ratios of 5:1 or greater, pH and contact time did not appear to be significant factors. Again, however, these results must be viewed with some caution due to the cyanide analytical problem mentioned previously (also see Section V of this report).

The results for destruction of total phenol (4AAP) by ozonation are also presented in Table VIII-78. These results do not reveal a definite trend, although the most effective removal was indicated at the highest ozone dosages, i.e., 8 and 24 mg

To summarize, the treatability study conducted at Mill 2122 demonstrated the effectiveness of polishing technologies (i.e., secondary settling or dual-media filtration) for removal of suspended solids when the initial TSS concentration was in the range of 30 to 50 mg/l. Much higher TSS concentrations often occur in the tailing pond decant at Mill 2122 due to the manner in which the pond is operated and effluent is withdrawn. Under conditions of high TSS loading, secondary settling with the use of a flocculating aid (i.e., polymer) would be a more practical effluent polishing technology than filtration. Lime addition provided little benefit. However, the effective removal of TSS by the polishing technologies also resulted in effective removal of metals. Cyanide was apparently destroyed by both hypochlorite and ozone. At high dosages of these oxidants, total phenol (4AAP) were also removed.

CONTROL AND TREATMENT PRACTICES

Control and Treatment of Wastewater at Placer Mines

Placer mining sites generally have limited area available for construction of treatment facilities. In addition, the lifetime of a given mining site is generally very short (1 to 5 years). However, as mining methods improve and economics of gold recovery become more favorable, the same area may be remined several times by different miners. The BPT effluent limitations governing the placer mining of gold were reserved because of insufficient economic data (Reference 68). A discussion of control practices at placer mines using gravity separation processes is presented here.

Placer mining consists of excavating waterborne or glacial deposits of gold bearing gravel and sands which can be separated by physical means. This separation is classified as gravity separation milling (reference Section IV). Since many placer deposits are deeply buried, bulldozers, front-end loaders, and draglines are being used for overburden stripping, sluicebox loading, and tailing removal operations. However, where water availability and physical characteristics permit, dredging or hydraulic methods are often favored based on cost.

Gold has historically been recovered from placer gravels by purely physical means. Gravity separation is accomplished in a sluicebox. Typically, a sluicebox consists of an open box to which a simple rectangular sluiceplate is mounted on a downward incline. A perforated metal sheet is fitted onto the bottom of the loading box, and riffle structures are mounted on the bottom of the sluiceplate. These riffles may consist of wooden strips, or steel or plastic plats which are angled away from the direction of flow in a manner designed to create pockets and eddy currents for the collection and retention of gold.

During actual sluicing operations, pay gravels (i.e., goldbearing gravels) are loaded into the upper end of the sluicebox and washed down the sluiceplate with water, which enters at right angles to (or against the direction of) gravel feed. Density differences allow the particles of gold to settle and become entrapped in the spaces between the riffle structures, while the less dense gravel and sands are washed down the sluiceplate. Eddy currents keep the spaces between riffle structures free of sand and gravel, but are not strong enough to wash out the gold.

Wastewater from placer mining operations consists primarily of the process water used in the gravity separation process. Recovery of placer gold by physical methods generally involves no crushing, grinding, or chemical reagent usage. As a result, the primary waste parameters requiring removal are the suspended and/or settleable solids generated during washing (i.e., sluicing, tabling, etc.) operations.

Arsenic is present at relatively high concentrations in some of the sediments being mined by placer methods. However, this arsenic occurs primarily in particulate form and can be removed by effective settling prior to discharge of the wash (sluice) water.

Current best treatment practice in this segment of the industry is the use of a dredge pond or a sedimentation pond. In some instances, the discharge of wastewater through old tailings achieves a filtering effect. The treatment effectiveness achieved by selected placer mining operations using this technology is indicated in Table VIII-79. Data provided here are documented in Reference 69, "Evaluation of Wastewater Treatment Practices Employed at Alaskan Gold Placer Operations" (July 1979).

Most of the over 250 active placer mines are located in Alaska. Some have estimated the actual number of placer mines at over

500. EPA Region X has issued National Pollution Discharge Elmination System (NPDES) permits to many placer miners in the State of Alaska which identify required settling pond designor effluent limitations, as indicated by the following (excerpted from a Region X NPDES permit):

"a. Provide settling pond(s) which are designed to contain the maximum volume of process water used during any one day's operation. Permittee shall design single and/or multiple ponds with channeling, diversions, etc., to enable routing of all uncontaminated waters around such treatment systems and also to prevent the washout of settling ponds resulting from normal high water runoff. Choice of this alternative requires no monitoring."

or

"b. Provide treatment of process wastes such that the following effluent limitations be achieved. The maximum daily concentration of settleable solids from the mining operation shall be 0.2 milliliter of solids per liter of effluent. This shall be measured by subtracting the value of settleable solids obtained above the intake structure from the value of settleable solids obtained from the effluent stream."

Few (if any) placer mining operations have ponds "which are designed to contain the maximum volume of process water used during any one day's operation." The actual retention capacity of the few existing settling ponds or pond systems at placer mining operations is typically two hours or less. However, as indicated in Table VIII-79, many of the operations which have installed settling ponds are producing an effluent which contains less than 1.0 ml/l/hr of settleable solids. Reductions of suspended solids attained at the operations are highly variable because of different flows, particle size distribution, working hours per day, etc.

Two practices were identified at placer mining operations. The first is the use of any screening device which effectively classifies (size separation) the paydirt prior to washing. The second is the use of multiple settling ponds.

The practice of screening greatly reduces the volume of water required for washing by eliminating the need for great hydraulic force to move large rocks and boulders through the sluice box. This increases retention time and improves settling conditions within a given settling pond by reducing the volume of wastewater requiring treatment.

The use of a number of smaller ponds in series appears to be more effective than a single larger pond at any given site. Generally, a limited area is available to placer miners for construction of a settling pond. As a result, ponds are generally small in size relative to the volume of wastewater to be treated. Therefore, it is not unusual that these ponds are severely short circuited. For this reason, the use of a number of small ponds in series serves to reduce hydraulic surges, offset short circuiting and reduce the velocity of flow, thereby improving conditions for removal of settleable solids. Also, some miners use sluice box tailings to construct dikes between several ponds in series. In passing from one pond to another, the wastewater must filter through these dikes. This practice provides very effective removal of settleable solids in most instances.

Multiple-settling pond systems have been used at placer Mines 4114, 4133, 4136, 4138, 4139, 4140, and 4141. Screening devices to classify paydirt prior to washing or sluicing have been used at placer mines 4133, 4136, 4138, and 4141. As indicated in Table VIII-79, all of the placer mines which employ multiple ponds and screening were capable of producing a treated effluent having less than 1.0 ml/l/hr of settleable solids. Mine 4142 also employs two ponds in series; however, these ponds were being short circuited and, as a result, were not as effective as they could have been.

A report prepared for the State of Alaska, Placer Mining Wastewater Settling Pond Demonstration Project, confirms that in theroy and practice, for settling placer mine wastewater discharges, an effective holding time of four hours of quiescent settling will reduce settleable solids to below detectable levels. For a pond to provide the equivalent of four hours quiescent settling, the pond generally must be designed for a holding time of more than four hours.

Control of Mine Drainage

It is a desirable practice to minimize the volume of water contaminated in a mine because the volume to be treated will be less. Best practices for mine drainage control result from careful planning and assessment of all phases of mining operations. Mining techniques used, water infiltration control, surface water control, erosion control, and regrading and revegetation of mined land are all essential considerations when planning for mine drainage control. In the past, inadequate planning resulted in a significant adverse impact on the environment due to mining. In many instances, extensive and costly control programs were necessary.

The types of mining operations (planned or existing) used to recover metal ores differ in many respects from those of the coal mining industry. This is important to note when considering the information available on mine drainage control in these industries. Mine drainage problems in the coal industry appear to be more widespread than those in the metal ore mining and dressing category. This is primarily because of the number of mines involved, geographic location, age, disturbed area, and geology of the mined areas. There is an abundance of literature describing the problem of mine drainage from both active and abandoned coal mines. The discussions which follow present the limited available information on mine drainage control in metal ore mines. However, references to practices employed in coal mining operations which may be applicable to metal ore mining are also presented.

Water-Infiltration Control

Diversion of water around a mine site to prevent its contact with possible pollution forming materials is an effective and widely applied control technique. Flumes, pipes, ditches, drains, and dikes are used in varying combinations, depending on the geology, geography, and hydrology of the mine area. This technique can be applied to many surface mines and mine waste piles.

Regrading, or recontouring, of some types of surface mines, and surface waste pile can be used to modify surface runoff, decrease erosion, and/or prevent infiltration of water into the mine area. There are many techniques available, but they are highly dependent on the geography and hydrology of the land and the availability of cover or fill materials. This practice, along with the establishment of a stable vegetative cover, is currently being used experimentally at one eastern metal ore mine to decrease erosion and stabilize soil on an abandoned waste pile. Use of regrading techniques at the larger open-pit mines may be limited only to the disturbed area surrounding the pit or to stabilization of some steep slopes.

Mine sealing techniques and procedures for sealing boreholes and fracture zones are more frequently applied to inactive or abandoned mines. Internal sealing by the placement of barriers within an underground mine can be used in an active mine with caution. Mine sealing practices are used either to prevent water from entering a mine or to promote flooding of an abandoned mine to decrease oxidation of pyritic materials. No data on the use or efficiency of mine sealing techniques in the metal ore mining and dressing industry were available for use in this report.

Control Practices in the Ore Mining and Dressing Industry

Most of the metal-ore mines examined in this report (both underground and open-pit) practice some measure of mine drainage control. These practices involve controlled pumping of mine drainages and application of a variety of treatment technologies, or use in a mill process. Use of mine water as makeup water in mill circuits is a desirable management practice and is widely implemented in this industry. In many areas of the West, water availability is limited, and water conservation practices are essential for mine/mill operations. Mine water which has been adequately treated is suitable for discharge to surface waters, and this practice is also common to this industry.

Regrading and revegetation of areas disturbed by mining is practiced at some operations, but is primarily directed at stabilization of tailing areas and, in some instances, waste or overburden piles. Documentation of the use and effectiveness of these practices is limited to uranium mining at this time.

Prevention or Control of Seepage from Treatment Ponds

Uranium mill wastewater is characterized by very high salinity and the presence of radioactive parameters. Therefore, at least four western states either have requirements or are developing requirements for seepage control to protect limited groundwater supplies (Reference 70).

Under certain conditions, unlined tailing or settling ponds may represent an acceptable level of environmental control for disposal of uranium mine water or milling wastewater (Reference 70). With proper siting, ponds could, in some instances, be located to take advantage of the properties of native soils in mitigation of the adverse effects of seepage. Many uranium deposits and milling facilities, however, are not located where the natural soils provide sufficient uptake of waterborne pollutants and prevention of contamination of groundwater.

Seepage rates and soil uptake of pollutants depends on the soil's chemical and physical properties, the design and construction of the pond itself, and the geological conditions prevalent at each site. Unlined ponds are best used under the following circumstances:

1. Deep groundwater table and/or soils exhibiting permeabilities sufficiently low to minimize the volume of seepage,

2. Native soils with significant capacity to remove and fix pollutants from seepage,

3. Arid climates, and

4. Geological and hydrological conditions at the site precluding contamination of aquifers or other bodies of water which are useful as water supplies.

The seepage rates from unlined uranium mill ponds depend upon the characteristics of the tailings, soils, underlying geoglogy, and hydrologic conditions prevalent at the site. Soils and tailing deposits exhibiting high permeabilities may permit high seepage rates, especially if sandy soils underlie the pond. If ponds are located on soils containing high proportions of natural clay or on impervious rock (such as shale), seepage rates can be reduced substantially. Permeability values as low as 10^{-6} to 10^{-8} cm/sec (down to 0.006 gal/min/acre) can often be achieved under these circumstances.

Pond Liner Technology

Prevention of seepage from impoundment systems can be achieved by the use of liners. Pond liners fall into two general categories: natural (clay or treated clay) and synthetic (commonly, polyvinyl chloride (PVC), polyethylene (PE), chlorinated polyethylene (CPE), or Hypalon).

Pond liners installed to date have usually been in new ponds which are used only for evaporating mill wastewater. Lining of tailing disposal ponds has not been practiced to a great extent for the following reasons:

1. Tailing ponds are usually larger than evaporation ponds. Large investments must be made for lining tailing ponds. The cost for lining a tailing pond may account for 60 to 90 percent of the pond capital cost. Where liners are installed and seepage is prevented, pond surface area must be increased in order to evaporate the wastewater;

2. Thicker liners may be required for tailing ponds than for evaporation ponds;

3. Reliable information on the long term performance of liners in tailing pond applications is lacking; and

4. Tailings themselves often prevent seepage as they are deposited in the ponds.

Natural (Clay) Liners. Clays can be effectively used in sealing ponds because of a layered structure and the ability of certain clay minerals to exchange cations with wastewater seeping through. Some clays, usually commercially identified as bentonite (montmorillonite), absorb water molecules between layers, resulting in a swelling of the clay structure. Under confined conditions, such as the case of a pond liner, swelling will be retarded, but the clay particles will be pressed tightly together. The amount of space between the particles is reduced, resulting in a decrease in permeability. The water which does permeate the clay will lose cations by ion exchange, preventing these contaminants from seepage into the groundwater.

According to Reference 70, the effective use of untreated clays for seepage control is limited to situations where the liner will be in contact with relatively fresh water. If high levels of dissolved salts, strong acids, or alkalies come into contact with the clay, ion exchange reactions between the wastewater and the clay will take place. This may remove some of the heavy metal ions present, but a loss of exchangeable ions from the clay results in a reduction of the swelling capacity of the clay and in eventual failure of the seal.

To improve the swelling characteristics of natural clays and make them more effective as pond liners, a clay may be treated with polymeric materials. The use of treated clay improves the sealing properties of the clay, and also permits a reduction in the amount required compared to untreated clay. Although complete containment of pond wastewater cannot be obtained with clay liners, permeabilities as low as 10^{-6} to 10^{-8} cm/sec (0.6 to 0.006 gal/min/acre) are achievable with treated clays (Reference 70).

Clay liners have the advantages of being easy to install with commonly used machinery, and they are relatively inert to most chemical constituents. One supplier of treated clay claims its product is effective in sealing ponds containing up to 20 percent dissolved solids. This product may be used in constructing a liner for a new pond or may be used to control lateral seepage through use of a slurry trench technique.

Commerical experience with treated clay liners is minimal. Mill 9446 uses a treated clay (variety unknown) to mitigate pond seepage. Plans call for American Colloid Company (Skokie, Illinois) to install a treated clay liner for a uranium project in Colorado (Reference 70).

Synthetic Pond Liners. Synthetic pond liners may also be used to control seepage from uranium mill ponds. These types of liners have an advantage over natural clay or treated clay liners, because they possess much lower permeability values than polyester reinforced Hypalon liners). Flexible synthetic liners, however, exhibit several disadvantages also:

1. Performance is highly dependent upon the quality of the foundation and substrate. Structural failures may result from poor initial design, poor substrate compaction, seismic disturbances, water buildup beneath the liner, inappropriate liner choice, or poor installation technique;

2. They are susceptible to degradation due to the chemical environment or exposure to the elements; and

3. They are more prone to puncture and tear during installation and may pose difficulties in field handling.

The most common synthetic liners used in the uranium industry for pond seepage control are PVC, PE, CPE, and Hypalon (Reference 69). They are used, alone or in conjunction, in thicknesses of 0.25 to 1.5 mm (10 to 60 mils). Different materials exhibit varying degrees of strength, flexibility, weatherability, and resistance to chemical attack.

No data are available on the long term performance of synthetic liners in uranium mill pond applications. However, tests conducted on these liners in sanitary landfill applications indicate little loss of tensile strength or tear or puncture resistance. Some increase in permeability has been noted, with the thermoplastic types (CPE, PVC, and Hypalon) tending to swell and soften.

Four currently operating uranium mills in the United States are using synthetic liners for seepage control. Mills 9422 and 9456 use Hypalon liners with thicknesses of 1.5 and 0.91 mm (60 and 36 mil), respectively. Mills 9402 and 9404 have decant pond liners constructed of 0.25 mm (10 mil) PVC bottoms and 0.51 mm (20 mil) CPE dike slopes. The PVC/CPE liners carry 15- and 10-year warranties, respectively.

It is common practice to cover synthetic liners with a layer of native soil to protect the liner from sunlight and to prevent damage to the liner from earthmoving equipment, if and when the pond requires dredging. Heavy-duty liners are preferred in uranium mill tailing pond applications, because they minimize posible mechanical damage when the pond is cleaned, generally have longer life and better aging properties, and are more resistant to the rocky soils present at many uranium mill sites (Reference 70).

Recently, two secured landfills in New York State and one in Ohio have been constructed for disposal of hazardous industrial wastes (Reference 71). These secured landfills make use of two layers of low permeability, natural clay liners, with a synthetic liner (reinforced Hypalon) placed in between. The Ohio facility is also equipped with an external, underdrain-type, leachate collection and monitoring system. Although this type of installation is very expensive, it represents state-of-the-art of liner technology. Where disposal of toxic liquids or solid wastes is necessary or groundwater supplies must be protected, this approach may represent the only viable alternative.

Other Seepage Control Methods

Other methods for mitigating seepage from uranium-mill ponds have been used successfully in the United States and Canada. These methods have been used to control both underseepage from mill ponds and lateral seepage through tailing dams or permeable subsoils.

Underseepage from an existing tailing pond at Mill 9401, located in New Mexico, is being controlled by collection and recycle of contaminated groundwater. Wells in the collection system pump groundwater contaminated by pond seepage from 12- to 18-meter (40- to 60-foot) depths back to the pond. Downgradient of the collection wells, injection wells are used to pump well water to dilute groundwater which might be contaminated by uncollected pond seepage (Reference 70). A system such as this can be effective only when specific favorable subsurface conditions prevail.

It is common practice to collect lateral seepage through existing tailing dams in a catch basin or sump for subsequent disposal or return to the pond (Reference 70). Visits to a number of facilities as part of this study, as well as visits during previous efforts, indicate that at least seven other mill facilities practice some form of seepage collection. The system in existence at uranium Mill 9401 is described above. Uranium Mill 9402 (also located in New Mexico) collects seepage from the tailing pond in a dam toe pond. Seepage occasionally appearing in an adjacent arroyo, due to precipitation events, is collected and pumped back to the pond.

Uranium Mill 9405 (an acid-leach facility) located in Colorado, collects seepage from its tailing pond and overflow from yellowcake precipitation thickeners and treats the combined waste stream to remove radium 226 and TSS prior to discharge.

Copper Mill 2121 collects seepage from its tailing pond and conveys it to a secondary settling pond, from which it is discharged. Lead/zinc Mill 3103, located in Missouri, also collects seepage and discharges it into a secondary settling pond. Mill 3123, located in Missouri, collects seepage at the toe of the tailing dam and pumps it back to the tailing pond.

Gold Mill 4101 intercepts seepage in a collection sump and pumps it back to the mill for reuse. This facility does not discharge to surface waters. Gold Mill 4105 has recently designed and installed a seepage collection system which takes seepage from the base of its tailing dam and pumps a volume in excess of 0.76

The use of lateral seepage control methods, such as the slurry trench technique, can be most effective when an impermeable layer exists beneath the impoundment. Otherwise, lateral seepage may escape containment by migrating through permeable materials under the dam. The lateral seepage curtain should extend down to the impermeable layer.

Lateral seepage of tailing pond water through the subsoil at a uranium mill in Eastern Ontario, Canada, is controlled by a grout curtain constructed of clay, bentonite, and cement (Reference 70). The grout slurry was injected into the subsurface alluvium, down to an impervious bedrock layer, and forms an underground barrier to the lateral flow of seepage to a nearby recreational lake. Monitoring the concentrations of dissolved radium 226 in the groundwater has demonstrated the effectiveness of the curtain. The effectiveness of this method is attributed to increased flow-path length and ion exchange with the montmorillonite clay in the grout.

TABLE VIII-1. ALTERNATIVES TO SODIUM CYANIDE FOR FLOTATION CONTROL

ORE	pH LEVEL	DEPRESSANT	DEPRESSANT USAGE*		PERCENTAGE RECOVERY			PERCENTAGE DEPRESSION		DEPRESSANT COST	
			kg/metric ton ground ore	lb/short ton ground ore	copper	lead	zinc	iron	zinc	per metric ton ground ore	per short ton ground ore
copper with pyrite	natural	Sodium cyanide	0.196	0.392	62.0	NA	NA	84.1%	NA	$0.18	$0.16
		Sodium sulfite	0.504	1.008	56.6	NA	NA	81.9	NA	$0.14	$0.13
	10 to 12	Sodium cyanide	0.196	0.392	60.9	NA	NA	82.6	NA	$0.18	$0.16
		Sodium sulfite	0.006	0.013	65.4	NA	NA	69.5	NA	< $0.01	< $0.01
		Sodium monosulfide	0.312	0.624	67.5	NA	NA	66.4	NA	$0.11	$0.09
copper/lead/zinc with pyrite	natural	Sodium cyanide	0.196	0.392	73.6	78.8	NA	85.7	77.8	$0.18	$0.16
		Sodium sulfite	0.504	1.008	74.6	74.9	NA	87.3	82.9	$0.14	$0.13
	10 to 12	Sodium cyanide	0.196	0.392	58.2	75.0	NA	87.4	77.2	$0.18	$0.16
		Sodium sulfite	0.504	1.008	74.2	80.3	NA	83.9	72.7	$0.14	$0.13
		Sodium monosulfide	0.003	0.006	73.4	75.9	NA	84.2	53.3	<$0.01	<$0.01
zinc with pyrite	natural	Sodium cyanide	0.196	0.392	NA	NA	7.97	93.3	NA	$0.18	$0.16
		Sodium sulfite	0.005	0.010	NA	NA	11.6	91.2	NA		
	10 to 12	Sodium cyanide	0.002	0.004	NA	NA	9.11	92.6	NA		
		(none)	–	–	NA	NA	11.2	91.9	NA	–	–
		Sodium monosulfide	0.003**	0.006	NA	NA	13.1	91.4	NA	< $0.01	< $0.01
		Sodium sulfite	0.504	1.008	NA	NA	12.9	91.2	NA	$0.14	$0.13

Based on References 37 and 38.

NA = not applicable

* Based on laboratory-scale experiments using 500 grams of ground ore.

**0.312 kg/metric ton ground ore not studied

TABLE VIII-2. RESULTS OF LABORATORY TESTS OF CYANIDE DESTRUCTION BY OZONATION AT MILL 6102

pH	Final Cyanide Concentration (mg/l)
5.2	0.36
7.4	0.09
8.1	0.06
9.3	0.05
9.4	0.04
10.2	0.03
11.4	0.02
12.7	0.04

Based on ozone dosage = 10 times stoichiometric;
15-minute contact time; and initial cyanide concentration of 0.55 mg/l.

TABLE VIII-3. RESULTS OF LABORATORY TESTS AT MILL 6102 DEMONSTRATING EFFECTS OF RESIDENCE TIME, pH, AND SODIUM HYPOCHLORITE CONCENTRATION ON CYANIDE DESTRUCTION WITH SODIUM HYPOCHLORITE

pH:	8.8		10.6		11.0	
NaOC1 Concentration:	10 mg/l	20 mg/l	10 mg/l	20 mg/l	10 mg/l	20 mg/l
Residence Time:						
30 minutes	-	0.08	0.04	0.03	0.03	0.01
60 minutes	-	0.05	0.03	0.02	0.03	0.02
90 minutes	-	0.07	0.04	0.02	0.03	0.02

Initial cyanide concentration was 0.19 mg/l.

TABLE VIII-4. EFFECTIVENESS OF WASTEWATER-TREATMENT ALTERNATIVES FOR REMOVAL OF CHRYSOTILE AT PILOT PLANTS

TREATMENT METHOD	FIBER CONCENTRATION (fibers/liter)	
	Raw Water	Treated Water
Sedimentation	4×10^{12}	5×10^{11}† to 1×10^{11}**
Sedimentation plus Mixed-Media Filtration*	4×10^{12}	1×10^{9}
Sedimentation plus Uncoated-Diatomaecous-Earth Filtration	4×10^{12}	3×10^{6}
Sedimentation plus Alum-Coated-Diatomaceous-Earth Filtration	4×10^{12}	1×10^{5}

Source: Reference 44

*Dual-media filtration with a column containing 25 mm (1 in.) of anthracite and 320 mm (12.5 in.) of graded sand.

†After 1 hour of sedimentation

**After 24 hours of sedimentation

TABLE VIII-5. EFFECTIVENESS OF WASTEWATER-TREATMENT ALTERNATIVES FOR REMOVAL OF TOTAL FIBERS AT ASBESTOS-CEMENT PROCESSING PLANT

TREATMENT METHOD	FIBER CONCENTRATION (fibers/liter)	
	Raw Water	Treated Water
Sedimentation (for 24 hours)	5×10^{9}*	9.3×10^{9}†
Sedimentation (for 24 hours) plus Sand Filtration	5×10^{9}*	3.2×10^{9}**

Source: Reference 44

Corresponding turbidities are:

*620 JTU's
† 1.0 JTU's
** 0.38 JTU's

TABLE VIII-6. EFFECTIVENESS OF WASTEWATER-TREATMENT ALTERNATIVES FOR REMOVAL OF TOTAL FIBERS AT ASBESTOS, QUEBEC, ASBESTOS MINE

TREATMENT METHOD	FIBER CONCENTRATION (fibers/liter)	
	Raw Water	Treated Water
Mixed-Media Filtration	1×10^9	3×10^7
Uncoated-Diatomaceous-Earth Filtration	1×10^9	3×10^6
Coated-Diatomaceous-Earth Filtration	1×10^9	8×10^4

Source: Reference 44

TABLE VIII-7. EFFECTIVENESS OF WASTEWATER-TREATMENT ALTERNATIVES FOR REMOVAL OF TOTAL FIBERS AT BAIE VERTE, NEWFOUNDLAND, ASBESTOS MINE

TREATMENT METHOD	FIBER CONCENTRATION (fibers/liter)	
	Raw Water	Treated Water
Sedimentation	1×10^9 (1×10^{11})	1×10^9 (1×10^{10})
Sedimentation plus Dual-Media Filtration	1×10^9 (1×10^{11})	1×10^8 (1×10^9)
Sedimentation plus Uncoated-Diatomaceous-Earth Filtration	1×10^9	2×10^6
Sedimentation plus Alum-Coated-Diatomaceous-Earth Filtration	1×10^9	$< 1 \times 10^5$

Source: Reference 44

Parentheses enclose results for a second sample.

TABLE VIII-8. COMPARISON OF TREATMENT-SYSTEM EFFECTIVENESS FOR TOTAL FIBERS AND CHRYSOTILE AT SEVERAL FACILITIES SURVEYED

FACILITY	TYPE OF SAMPLING	INFLUENT CONC. (fibers/liter)		EFFLUENT CONC. (fibers/liter)		TREATMENT REDUCTION FACTOR	
		TOTAL FIBERS	CHRYSOTILE	TOTAL FIBERS	CHRYSOTILE	TOTAL FIBER	CHRYSOTILE
4401 (Mine-Water Settling Pond)	S	3.8×10^7	1.1×10^7	5.7×10^7	1.1×10^6	–	10
4401 (Tailing Pond)	S	7.1×10^{11}	1.1×10^{11}	2.1×10^9	1.8×10^8	$>10^2$	$\sim 10^3$
5102 (Mine-Water Treatment System)	S	3.5×10^7	5.5×10^6	1.4×10^9	2.0×10^8	INC	INC
5102 (Mine-Water Treatment System)	V	3.6×10^7	5.5×10^6	1.0×10^8	3.3×10^6	INC	–
2122 (Tailing Pond)	S	2.5×10^{12}	4.3×10^{11}	4.3×10^9	6.7×10^8	$\sim 10^3$	$\sim 10^3$
2122 (Tailing Pond)	V	ND	ND	6.3×10^9	$<2.2 \times 10^5$	–	–
2122 (Tailing Pond)	S	6.1×10^{12}	6.2×10^{11}	3.7×10^7	8.2×10^6	$\sim 10^5$	$\sim 10^5$
2122 (Tailing Pond)	V	ND	ND	2.7×10^8	$<2.2 \times 10^5$	–	–
2122 (Tailing Pond)	V	ND	ND	1.3×10^7	9.1×10^5	–	–
2121 (Treatment System)	S	3.0×10^{11}	5.5×10^{10}	8.2×10^6	5.5×10^5	10^4-10^5	10^5
2120 (Tailing Pond)	S	1.2×10^{12}	3.1×10^{11}	1.2×10^9	3.0×10^8	10^3	10^3
2120 (Tailing Pond)	V	1.3×10^{13}	1.7×10^{12}	7.8×10^7	1.2×10^7	10^5-10^6	$\sim 10^5$
2120 (Mine-Water Treatment System)	S	4.6×10^7	1.8×10^6	7.2×10^7	8.2×10^6	–	–
2117 (Treatment Plant)	S	2.5×10^8	5.5×10^6	3.4×10^6	$<2.2 \times 10^5$	$\sim 10^2$	>10

S = screen sampling
V = verification phase
INC = increase
ND = no data

TABLE VIII-8. COMPARISON OF TREATMENT-SYSTEM EFFECTIVENESS FOR TOTAL FIBERS AND CHRYSOTILE AT SEVERAL FACILITIES SURVEYED (Continued)

FACILITY	TYPE OF SAMPLING	INFLUENT CONC. (fibers/liter)		EFFLUENT CONC. (fibers/liter)		TREATMENT REDUCTION FACTOR	
		TOTAL FIBERS	CHRYSOTILE	TOTAL FIBERS	CHRYSOTILE	TOTAL FIBER	CHRYSOTILE
2117 (Treatment Plant)	S	1.6×10^7	$< 6.8 \times 10^5$	8.8×10^6	5.5×10^5	<10	–
2117 (Tailing Pond)	S	1.9×10^{11}	5.5×10^{10}	9.2×10^6	4.4×10^5	10^4-10^5	10^5
1105 (Mine-Water Settling Pond)	S	1.6×10^7	3.8×10^6	4.2×10^7	3.8×10^6	–	–
1108 (Tailing Pond)	S	2.3×10^{11}	3.8×10^{11}	4.3×10^7	4.1×10^6	$\sim 10^4$	$\sim 10^5$
9202 (Tailing Pond)	S	1.2×10^{12}	1.5×10^{11}	7.7×10^8	5.7×10^7	10^3-10^4	10^3-10^4
6101 (Tailing Pond)	S	3.8×10^{11}	2.7×10^{10}	3.3×10^{10}	2.0×10^9	10	10
6101 (Tailing Pond)	V	5.8×10^{11}	2.7×10^{11}	8.7×10^7	9.7×10^6	$\sim 10^4$	10^4-10^5
3107 (Treatment System)	S	2.2×10^{10}	1.4×10^9	4.1×10^8	4.1×10^7	$\sim 10^2$	$\sim 10^2$
3121 (Tailing Pond)	S	1.8×10^{11}	2.2×10^{10}	1.6×10^9	$< 3.3 \times 10^5$	10^2	10^5
3103 (Tailing-Settling Pond)	V	2.1×10^{11}	8.2×10^{10}	9.9×10^6	1.1×10^6	10^4-10^5	10^4
3101 (Tailing-Settling Pond)	V	2.4×10^{10}	3.2×10^9	1.9×10^7	2.7×10^6	10^3	10^3
3110 (Tailing Pond)	S	9.0×10^{11}	2.6×10^{11}	3.4×10^8	2.4×10^7	10^3	10^4
9905 (Settling Pond)	S	7.1×10^9	1.1×10^9	1.5×10^8	1.3×10^6	10	$\sim 10^3$

S = screen sampling
V = verification phase
INC = increase
ND = no data

TABLE VIII-8. COMPARISON OF TREATMENT-SYSTEM EFFECTIVENESS FOR TOTAL FIBERS AND CHRYSOTILE AT SEVERAL FACILITIES SURVEYED (Continued)

FACILITY	TYPE OF SAMPLING	INFLUENT CONC. (fibers/liter)		EFFLUENT CONC. (fibers/liter)		TREATMENT REDUCTION FACTOR	
		TOTAL FIBERS	CHRYSOTILE	TOTAL FIBERS	CHRYSOTILE	TOTAL FIBER	CHRYSOTILE
9402 (Mine-Water Treatment System)	S	1.2×10^8	5.2×10^6	4.3×10^8	5.3×10^7	–	INC
9408 (Mine-Water Treatment System)	S	1.6×10^9	1.9×10^8	2.3×10^9	2.0×10^8	–	–
9408 (Mine-Water Treatment System)	V	1.4×10^8	1.5×10^7	7.3×10^6	$<2.2 \times 10^5$	$10 \cdot 10^2$	$\sim 10^2$
9405 (Mill Settling Pond)	S	2.9×10^8	2.3×10^7	1.2×10^9	1.5×10^8	INC	INC
9405 (Mill Settling Pond)	V	1.0×10^8	$<2.2 \times 10^5$	6.6×10^7	$<2.2 \times 10^5$	<10	–
9411 (Mine-Water Treatment System)	S	2.3×10^9	1.1×10^8	5.7×10^8	2.7×10^7	<10	<10
6104 (Mine-Water Treatment System)	S	7.7×10^6	$<2.1 \times 10^5$	3.3×10^7	8.2×10^6	INC	INC

S = screen sampling
V = verification phase
INC = increase
ND = no data

TABLE VIII-9. EFFLUENT QUALITY ATTAINED BY USE OF BARIUM SALTS FOR REMOVAL OF RADIUM FROM WASTEWATER AT VARIOUS URANIUM MINE AND MILL FACILITIES

OPERATION		AMOUNT (mg/l) OF $BaCl_2$ ADDED	RADIUM CONCENTRATIONS (picocuries/l)				PERCENTAGE OF RADIUM REMOVED	
			BEFORE $BaCl_2$ TREATMENT		AFTER $BaCl_2$ TREATMENT			
			Total	Dissolved	Total	Dissolved	Total	Dissolved
Mills	9403[1]	7.4[2]	111 (±1.1)	–	4.09 (±0.41)	–	96.3	–
	9405[1]	9.5	15.9 (±1.6)	–	<1.0	–	>93.7	–
	9405*[8]	9.5†	39.2 (±3.9)	33.3 (±3.3)	5.05 (±0.5)	<2	87.1	>93.9
Mines	9411	5	35.4 (±0.3)[4]	15.5 (±2.0)[5]	8.4 (±0.1)[4]	0.2 (±0.1)[5]	76.3	98.7
	9411*[9]	10	56.9 (±5.7)	60.2 (±6.0)	<2	–	>96	–
	9412[1,6]	10.4	48.9 (±0.2)	4.7 (±0.1)	10.9 (±0.2)	1.6 (±0.1)	77.7	66.0
	9408	55	123.6 (±1.5)[7]	37.7 (±0.3)[4]	2.1 (±0.23)[7]	0.6 (±0.1)[4]	98.3	98.4
	9408*	55	142 (±14)	120 (±12)	1.12 (±0.11)	<0.9	99	>99
	9452[3]	45	955	93.4	7.18	<1	99.2	>99

1. Data obtained from single grab sampling and analysis (April, 1976).
2. Calculated value based on average flow and annual $BaCl_2$ usage.
3. Includes ion exchange treatment; facility visited August 1978.
4. Data obtained from analysis of two grab samples (April, 1976).
5. Company data for February 1975 (Average of 12 grab samples).
6. Final discharge to dry watercourse.
7. Colorado Dept. of Health data for period January 1973 through February 1975 (Average of 24 samples analyzed for "extractable" Ra 226).
8. Data obtained from composite of two grab samples representing two separate influent points (May, 1977).
9. Note that the dosage has doubled apparently enhancing the treatment system efficiency.

* Updated data obtained during sampling trips occuring April-May, 1977. All samples, unless otherwise indicated, are 24-hr composites.

† Dosage rates are assumed to remain the same as previous rates.

( ) Parenthetical values indicate analytical accuracy.

TABLE VIII-10. RESULTS OF MINE WATER TREATMENT* BY LIME ADDITION AT COPPER MINE 2120

TREATMENT pH	TOTAL METAL CONCENTRATION (mg/l)						
	Fe	Cu	Zn	Pb	Cd	As	Hg
6.2	11.6	0.26	15.0	0.01	0.19	0.002	<0.0005
8.5	0.45	0.10	0.25	<0.01	0.02	0.006	0.0006
10.3	0.12	0.04	0.56	0.01	0.02	0.003	<0.0005
11.5	0.17	0.07	0.30	0.01	0.01	0.008	<0.0005
TREATMENT pH	DISSOLVED METAL CONCENTRATION (mg/l)						
	Fe	Cu	Zn	Pb	Cd	As	Hg
6.2	7.2	0.25	14.6	<0.01	0.19	–	–
8.5	0.05	0.03	0.14	<0.01	0.02	–	–
10.3	0.03	0.03	0.10	<0.01	0.02	–	–
11.5	0.03	0.04	0.23	<0.01	0.01	–	–

*Bench-scale experiments; raw data not provided; a measure of effectiveness is obtained by comparison to values shown at initial pH.

TABLE VIII-11. RESULTS OF COMBINED MINE AND MILL WASTEWATER TREATMENT* BY LIME ADDITION AT COPPER MINE/MILL 2120

SAMPLE	TREATMENT pH	TOTAL METAL CONCENTRATIONS (mg/l)						
		Fe	Cu	Zn	Pb	Cd	As	Hg
Mine water + mill tailings	6.5	0.35	1.09	22.8	0.01	0.32	0.006	0.0006
	7.0	0.06	0.33	5.4	<0.01	0.18	0.009	<0.0005
	8.0	0.05	0.06	0.29	<0.01	0.04	0.002	<0.0005
	9.0	0.05	0.04	0.09	0.01	0.02	0.004	<0.0005
	10.0	0.05	0.05	0.06	0.01	0.01	0.004	<0.0005
	11.0	0.10	0.04	0.04	<0.01	0.01	0.006	0.0008
Mill tailings (control)	–	0.06	0.09	0.06	0.01	0.01	0.003	<0.0005

SAMPLE	TREATMENT pH	DISSOLVED METAL CONCENTRATIONS (mg/l)						
		Fe	Cu	Zn	Pb	Cd	As	Hg
Mine water + mill tailings	6.5	0.06	1.00	22.1	<0.01	0.31	–	–
	7.0	0.03	0.26	5.4	<0.01	0.18	–	–
	8.0	0.02	0.06	0.29	<0.01	0.04	–	–
	9.0	0.02	0.04	0.09	<0.01	0.02	–	–
	10.0	0.02	0.05	0.06	<0.01	0.01	–	–
	11.0	0.02	0.03	0.04	<0.01	0.01	–	–
Tailings (control)	–	0.01	0.04	0.06	<0.01	<0.01	–	–

*Bench-scale experiments. 5 parts mine water:9 parts mill tailings.

Raw data not provided; a measure of effectiveness of treatment is obtained by comparison to values shown at initial pH.

TABLE VIII-12. RESULTS OF COMBINED MINE WATER + BARREN LEACH SOLUTION TREATMENT* BY LIME ADDITION AT COPPER MINE/MILL 2120

SAMPLE	TREATMENT pH	TOTAL METAL CONCENTRATIONS (mg/l)					
		Fe	Cu	Zn	Pb	Cd	As
Mine + barren leach water	6.1	533.0	0.68	150.0	0.01	1.48	0.007
	7.7	15.6	0.08	1.90	0.01	0.22	0.004
	9.8	0.10	0.04	0.12	0.01	0.01	0.004
	11.5	0.07	0.05	0.96	0.01	0.01	0.002
SAMPLE	TREATMENT pH	DISSOLVED METAL CONCENTRATIONS (mg/l)					
		Fe	Cu	Zn	Pb	Cd	As
Mine + barren leach water	6.1	532.0	0.68	150.0	0.01	1.47	–
	7.7	14.0	0.03	1.90	0.01	0.21	–
	9.8	0.10	0.03	0.12	0.01	0.01	–
	11.5	0.04	0.04	0.83	0.01	0.01	–

*Bench-scale experiments. 5 parts mine water:1.5 parts barren leach solution

Raw data not provided; a measure of effectiveness of treatment is obtained by comparison to values shown at initial pH.

TABLE VIII-13. RESULTS OF COMBINED MINE WATER + BARREN LEACH SOLUTION + MILL TAILINGS TREATMENT* BY LIME ADDITION AT COPPER MINE/MILL 2120

SAMPLE	TREATMENT pH	TOTAL METAL CONCENTRATIONS (mg/l)					
		Fe	Cu	Zn	Pb	Cd	As
Mine water + mill tailings + barren leach water	6.2	191.0	0.48	86.0	0.01	0.70	0.008
	7.0	75.7	0.08	18.0	0.01	0.33	0.004
	7.9	0.59	0.06	50.0	0.01	0.04	0.002
	9.3	0.14	0.05	0.08	0.01	0.01	0.002
	10.0	0.09	0.05	0.12	0.01	0.01	0.004
	11.1	0.05	0.04	0.05	0.01	0.01	0.002
Tailings (control)	10.6	0.03	0.04	0.05	0.01	<0.01	0.002

SAMPLE	TREATMENT pH	DISSOLVED METAL CONCENTRATIONS (mg/l)					
		Fe	Cu	Zn	Pb	Cd	As
Mine water + mill tailings + barren leach water	6.2	175.0	0.48	83.0	0.01	0.60	–
	7.0	68.2	0.08	17.2	0.01	0.31	–
	7.9	0.13	0.04	0.37	0.01	0.04	–
	9.3	0.07	0.05	0.07	0.01	0.01	–
	10.0	0.06	0.05	0.12	0.01	0.01	–
	11.1	0.03	0.04	0.05	0.01	0.01	–
Tailings (control)	10.6	<0.01	0.03	0.04	0.01	<0.01	–

*Bench-scale experiments. 5 parts mine water:1.5 parts barren leach solution:9 parts mill tailings.
Raw data not provided; a measure of effectiveness of treatment is obtained by comparison to values shown at initial pH.

TABLE VIII-14. CHARACTERISTICS OF RAW MINE DRAINAGE TREATED DURING PILOT-SCALE EXPERIMENTS IN NEW BRUNSWICK, CANADA

PARAMETER	CONCENTRATION (mg/l)					
	MINE 1		MINE 2		MINE 3	
	Mean	Range	Mean	Range	Mean	Range
pH*	2.6	2.4 to 3.2	2.7	2.3 to 2.9	3.0	2.8 to 3.3
Sulfate	10,100	1,860 to 14,892	4,454	2,354 to 7,290	1,121	729 to 1,790
Acidity (as $CaCO_3$)	6,511	4,550 to 9,650	4,219	2,600 to 7,000	746	70 to 1,530
Cu	10.0	4.8 to 22.3	47.2	24.3 to 76.0	19.4	1.0 to 52.0
Fe	1,534	8.5 to 3,211	718	350 to 1,380	77	24 to 230
Pb	3.9	0.9 to 10.3	1.2	0.3 to 3.2	1.3	0.1 to 5.0
Zn	1,158	142 to 1,615	538	390 to 723	114	18 to 185
Suspended Solids	172	70 to 645	65	10 to 190	31	5 to 90

*Value in pH units

Source: Reference 54

TABLE VIII-15. EFFLUENT QUALITY ATTAINED DURING PILOT-SCALE MINE-WATER TREATMENT STUDY IN NEW BRUNSWICK, CANADA

MINE	STREAM	EXTRACTABLE METAL (mg/l)			
		LEAD	ZINC	COPPER	IRON
COMPARISON OF EFFLUENT QUALITIES DURING ALL PERIODS OF STUDY – OPERATING PARAMETERS VARIED					
1	Clarifier Overflow	0.18 (0.05 to 0.39)	0.41 (0.13 to 0.87)	0.04 (0.02 to 0.10)	0.30 (0.14 to 0.65)
	Bucket-Settled	0.18 (0.08 to 0.25)	0.23 (0.20 to 0.25)	0.03 (0.01 to 0.05)	0.16 (0.08 to 0.29)
	Sand-Filtered	0.12 (0.07 to 0.15)	0.26 (0.14 to 0.38)	0.03 (0.02 to 0.04)	0.23 (0.09 to 0.63)
2	Clarifier Overflow	0.35 (0.05 to 0.62)	0.52 (0.07 to 1.42)	0.06 (0.03 to 0.19)	0.54 (0.12 to 2.51)
	Bucket-Settled	0.26 (0.01 to 0.50)	0.26 (0.03 to 0.60)	0.03 (0.02 to 0.07)	0.20 (0.04 to 0.41)
	Sand-Filtered	0.31 (0.01 to 0.50)	0.28 (0.03 to 0.58)	0.03 (0.02 to 0.04)	0.11 (0.02 to 0.20)
3	Clarifier Overflow	0.13 (0.05 to 0.62)	0.64 (0.14 to 1.45)	0.10 (0.01 to 0.30)	0.47 (0.09 to 1.40)
	Basin-Settled	0.11 (0.05 to 0.36)	0.29 (0.01 to 0.74)	0.05 (0.01 to 0.30)	0.26 (0.01 to 0.61)
	Sand-Filtered	0.10 (0.05 to 0.36)	0.21 (0.01 to 0.75)	0.04 (0.01 to 0.30)	0.22 (0.01 to 0.89)
COMPARISON OF EFFLUENT QUALITIES DURING PERIODS OF OPTIMIZED STEADY OPERATION					
1	Clarifier Overflow	0.18 (0.01 to 0.35)	0.33 (0.13 to 0.52)	0.04 (0.03 to 0.06)	0.19 (0.10 to 0.26)
	Bucket-Settled	0.21 (0.16 to 0.25)	0.29 (0.28 to 0.30)	0.04 (0.03 to 0.04)	0.18 (0.15 to 0.21)
	Sand-Filtered	0.15 (0.14 to 0.15)	0.39 (0.38 to 0.39)	0.03 (0.03 to 0.03)	0.20 (0.14 to 0.26)
2	Clarifier Overflow	0.44 (0.25 to 0.62)	0.45 (0.27 to 0.69)	0.05 (0.03 to 0.07)	0.42 (0.14 to 0.45)
	Bucket-Settled	0.29 (0.01 to 0.50)	0.22 (0.03 to 0.60)	0.03 (0.02 to 0.03)	0.17 (0.04 to 0.29)
	Sand-Filtered	0.29 (0.17 to 0.42)	0.15 (0.03 to 0.28)	0.03 (0.02 to 0.03)	0.13 (0.11 to 0.17)
3	Clarifier Overflow	0.15 (0.09 to 0.25)	0.35 (0.14 to 0.90)	0.06 (0.03 to 0.11)	0.26 (0.09 to 0.60)
	Basin-Settled	0.11 (0.05 to 0.18)	0.22 (0.13 to 0.40)	0.07 (0.02 to 0.15)	0.24 (0.10 to 0.30)
	Sand-Filtered	0.08 (0.05 to 0.22)	0.12 (0.03 to 0.18)	0.03 (0.02 to 0.04)	0.14 (0.08 to 0.23)

Source: Reference 54

TABLE VIII-16. RESULTS OF MINE-WATER TREATMENT BY LIME ADDITION AT GOLD MINE 4102

WASTE STREAM	pH	CONCENTRATION (mg/l) OF PARAMETER				
			Pb	Cu	Zn	Fe
Raw mine drainage	6.0	Total	2.2	0.02	9.8	40
	5.9	Dissolved	0.02	0.01	9.6	0.3
Treated mine water after adjustment of pH with lime	7.4	Dissolved	<0.01	0.01	3.84	0.4
	8.1	Dissolved	0.01	0.01	0.65	0.4
	9.2	Dissolved	<0.01	0.01	0.02	0.2

TABLE VIII-17. RESULTS OF LABORATORY-SCALE MINE-WATER TREATMENT STUDY AT LEAD/ZINC MINE 3113

UNIT TREATMENT PROCESSES EMPLOYED	EFFLUENT CONCENTRATIONS ATTAINED* (mg/l)				
	pH†	Cu	Pb	Zn	Fe
No treatment (control)	7.0 to 7.5	N.A.	N.A.	20	60
Lime addition to pH 10, sedimentation**	10	0.022 to 0.033	0.05 to 0.12	0.125 to 0.19	N.A.
Lime addition to pH 11, sedimentation**	11	0.22	0.05	0.095	N.A.

Results shown are based on jar tests.

*All metals concentrations are based on "total" analyses.

†Value in pH units

**Theoretical retention times were variable and not always specified.

N.A. = Not analyzed

TABLE VIII-18. EPA-SPONSORED WASTEWATER TREATABILITY STUDIES CONDUCTED BY CALSPAN AT VARIOUS SITES IN ORE MINING AND DRESSING INDUSTRY

SITE IDENTIFICATION	PERIOD OF STUDY	COMMENTS
Mine/Mill 3121 (Pb/ZN)*	August 3-13, 1978 March 19-29, 1979	Polishing Treatments (Filtration, Secondary Settling), Lime Precipitation and Cyanide Destruction Technology (Ozonation, Alkaline Chlorination) Investigated
Mine/Mill/Smelter/ Refinery 3107 (Pb/Zn)*	August 14-19, 1978	Polishing Treatments (Filtration, Secondary Settling) Investigated
Mill 2122 (Cu)*	September 5-15,1978 January 8-19, 1979	Polishing Treatments (Filtration, Secondary Settling), Lime Precipitation and Cyanide Destruction Technology (Ozonation, Alkaline Chlorination) Investigated
Mine 3113 (Pb/Zn)*	September 22-29, 1978	Lime Precipitation, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration Investigated
Mine 5102 (Al)	October 10-16, 1978	Polishing Treatments (Filtration, Secondary Settling) and Lime Precipitation Investigated
Mill 9401 (U)**	October 23-30, 1978	Bench-Scale and Pilot-Scale Investigation of IX, pH Adjustment with H_2SO_4, Ferrous Sulfate Coprecipitation, Barium Chloride Coprecipitation, Settling, and Filtration.
Mill 9402 (U)**	November 1-10, 1978 December 4-14, 1978	Treatment Scheme Employing Lime Addition, Aeration, Barium Chloride Addition, Settling, and Filtration Investigated

* Operations in base and precious metals subcategory

**Operations in uranium ore subcategory

TABLE VIII-19. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT LEAD/ZINC MINE/MILL 3121

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
		MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	75	7.8	6.7 - 9.1	n.a.	n.a.
TSS	13	4.5	1 - 10	↓	↓
Sb	↓	<0.5	<0.5	↓	↓
As	↓	0.001	<0.001-0.025	↓	↓
Be	↓	<0.002	<0.002	↓	↓
Cd	↓	0.002	0.005-0.011	↓	↓
Cr	↓	<0.01	<0.01	↓	↓
Cu	↓	0.10	0.02-0.16	↓	↓
Pb	↓	0.21	0.18-0.25	↓	↓
Hg	↓	0.0002	<0.0002-0.0005	↓	↓
Ni	↓	<0.02	<0.02	↓	↓
Se	↓	0.002	<0.002-0.004	↓	↓
Ag	↓	0.01	<0.01-0.05	↓	↓
T1	↓	<0.01	<0.01	↓	↓
Zn	↓	0.74	0.25-1.25	↓	↓

Based on observations made in period 6 through 10 August 1978.
n.a. = Not Analyzed.

TABLE VIII-20. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT LEAD/ZINC MINE/MILL 3121 DURING PERIOD OF MARCH 19-29, 1979

POLLUTANT PARAMETER	"TOTAL" CONCENTRATION (mg/l)	
	MEAN $\bar{x}$	RANGE
pH	8.9	8.8-9.1
TSS	10	5-14
Sb	<0.10	-
As	<0.0020	-
Be	<0.0050	-
Cd (total)	0.016	0.015-0.020
(diss.)	0.0070	<0.0050-0.0010
Cr	0.022	<0.020-0.030
Cu (total)	0.19	0.15-0.23
(diss.)	0.036	0.020-0.050
Pb (total)	0.22	0.11-0.30
(diss.)	0.024	<0.020-0.030
Hg	0.0005	<0.0005-0.0010
Ni	<0.020	-
Se	<0.0050	-
Ag	<0.020	-
T1	<0.10	-
Zn (total)	2.0	1.4-2.6
(diss.)	0.16	0.080-0.24
Fe (total)	0.55	0.24-0.78
(diss.)	0.022	<0.020-0.030
CN	0.079	0.040-0.125

TABLE VIII-21. OBSERVED VARIATION WITH TIME OF pH AND COPPER AND ZINC CONCENTRATIONS OF TAILING-POND DECANT AT LEAD/ZINC MINE/MILL 3121

DATE (1978)	POLLUTANT PARAMETER CONCENTRATION (mg/l)			COMMENTS
	pH*	Cu	Zn	
August 6	6.9 to 7.4	0.004	1.1	Mill not operating
August 7	6.9 to 7.7	0.05	1.2	Mill not operating
August 7	7.3 to 7.7	0.02	1.3	Mill not operating
August 8	6.7 to 7.7	0.08	0.94	Mill startup 4:30 PM, Aug. 7
August 8	7.6 to 7.8	0.08	0.75	Mill operating
August 8	7.4 to 7.6	0.09	0.76	Mill operating
August 8	7.2 to 7.6	0.11	0.79	Mill operating
August 8	7.1 to 7.4	0.14	0.80	Mill operating
August 9	8.2 to 8.3	0.14	0.55	Mill operating
August 9	8.2 to 8.3	0.12	0.44	Mill operating
August 10	8.5 to 8.6	0.16	0.49	Mill operating
August 10	8.5 to 8.9	0.14	0.28	Mill operating
August 10	8.6 to 9.1	0.13	0.25	Mill operating

*Value in pH units

TABLE VIII-22. SUMMARY OF TREATED EFFLUENT QUALITY ATTAINED WITH PILOT-SCALE UNIT TREATMENT PROCESSES AT MINE/MILL 3121 DURING AUGUST STUDY

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATION* ATTAINED (mg/l)				
	pH[†]	TSS	Cu	Pb	Zn
Secondary Settling (approx. 11- to 22-hr theoretical retention time)	8.2 - 8.5	3	0.11	0.10	0.24
Lime Addition to pH 9.2, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time)	9.2	17	0.05	0.08	0.38
Lime Addition to pH 9.2, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time), Filtration	9.2	1	0.02	0.04	0.16
Lime Addition to pH 11.3, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time)	11.3	n.a.	0.03	0.05	0.13
Lime Addition to pH 11.3, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time), Filtration	11.3	<1	0.02	0.06	0.08
Filtration	7.4** (6.7 to 7.8)	<1** (<1 to 2)	0.02** (<0.01 to 0.04)	0.09** (0.03 to 0.12)	0.61** (0.24 to 1.1)
Filtration	8.3** (7.7 to 9.1)	<1** (<1 - 1)	0.05** (0.03 to 0.06)	0.035** (0.01 to 0.06)	0.044** (0.02 to 0.06)

* All metals concentrations are based on "total" analyses

[†] Value in pH units

** Average concentrations attained

( ) Range of concentrations attained

n.a. = Not analyzed

TABLE VIII-23. SUMMARY OF TREATED EFFLUENT QUALITY ATTAINED WITH PILOT-SCALE UNIT TREATMENT PROCESSES AT MINE/MILL 3121 DURING MARCH STUDY.

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATIONS ATTAINED (mg/l)*				
	pH†	TSS	Cu	Pb	Zn
Filtration	8.9-9.0	7 (3-10)	0.14 (0.12-0.16)	0.12 (0.050-0.22)	1.4 (0.96-1.8)
Lime Addition to pH 10.5, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time)	10.5	24 (20-29)	0.13 (0.11-0.14)	0.12 (0.060-0.16)	1.0 (0.52-1.4)
Lime Addition to pH 10.5, Polymer Addition, Flocculation, Secondary Settling (approx. 2.6-hr theoretical retention time), Filtration	10.3	2 (<1-3)	0.053 (0.040-0.070)	0.040 (0.030-0.040)	0.26 (0.040-0.48)

† pH Units

* Values given are mean and range, in ( ), concentrations

**All metal concentrations are based on "total" analyses

TABLE VIII-24. RESULTS OF OZONATION FOR DESTRUCTION OF CYANIDE

Initial CN^- concentration = 0.11

O_3 Dosage mg/min	O_3 Dosage Ratio O_3/CN	Retention Time (min)	Final CN Concentration (mg/1)
1.4	10:1	10	0.066
0.48	10:1	30	0.080
0.36	10:1	45	0.081
0.7	5:1	10	0.108
0	-	10	0.115

TABLE VIII-25. EFFLUENT FROM LEAD/ZINC MINE/MILL/SMELTER/ REFINERY 3107 PHYSICAL/CHEMICAL-TREATMENT PLANT

PARAMETER	CONCENTRATION (mg/1)*	
	Average	Range
pH**	8.78	8.5 to 8.9
TSS	15	8.3 to 26
Cd	0.16	0.044 to 0.58
Pb	0.15	0.08 to 0.26
Zn	4.5	1.8 to 8.5
Hg	0.0033	0.0010 to 0.023

Based on industry data collected during the period of December 1974 through April 1977.

*All metals concentrations are based on "total" analyses

**Value in pH units

TABLE VIII-26. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT LEAD/ZINC MINE/MILL/SMELTER/REFINERY 3107

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l) MEAN $\bar{x}$	RANGE	"DISSOLVED" CONCENTRATION (mg/l) MEAN $\bar{x}$	RANGE
pH	64	8.5	8.1 - 8.7	n.a.	n.a.
TSS	11	16	13 - 20	↓	↓
Sb	12	<0.5	<0.5	↓	↓
As	↓	0.0024	0.0015-0.0030	↓	↓
Be	↓	<0.002	<0.002	↓	↓
Cd	↓	0.12	0.075 - 0.16	0.036	0.025-0.050
Cr	↓	0.010	<0.010 - 0.010	n.a.	n.a.
Cu	↓	0.031	0.020 - 0.045	0.021	0.020-0.030
Pb	↓	0.13	0.090 - 0.17	0.073	<0.02-0.17
Hg	↓	0.0006	0.0003-0.0012	n.a.	n.a.
Ni	↓	0.030	<0.02 - 0.060	↓	↓
Se	↓	0.002	<0.002-0.003	↓	↓
Ag	↓	<0.01	<0.01	↓	↓
T1	↓	0.012	0.010 - 0.025	↓	↓
Zn	↓	2.9	1.8 - 4.2	0.055	0.030-0.12

Based on observations made in period 14 through 19 August 1978.
n.a. = Not analyzed.

TABLE VIII-27. SUMMARY OF TREATED EFFLUENT QUALITY ATTAINED WITH PILOT-SCALE UNIT TREATMENT PROCESSES AT MINE/MILL/SMELTER/REFINERY 3107

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATION* ATTAINED (mg/l)						
	pH[†]	TSS	Cd	Cu	Pb	Hg	Zn
Secondary Sedimentation	7.8	3	0.065	0.020	0.080	n.a.	0.79
Polymer Addition, Secondary Sedimentation	8.1	6	0.060	0.015	0.070	n.a.	1.0
Filtration	8.5** (8.1 to 8.7)	<1** (<1)	0.035** (0.015 to 0.070)	0.016** (0.01 to 0.02)	0.061** (0.030 to 0.09)	n.a.	0.042** (0.015 to 0.080)

* All metals concentrations are based on "total" analyses
† Value in pH units
**Average concentrations attained
( ) Range of concentrations attained
n.a. Not analyzed

TABLE VIII-28. CHARACTER OF DRAINAGE FROM LEAD/ZINC MINE 3113

PARAMETER	CONCENTRATION (mg/1)*	
	Average	Range
pH**	4.2	2.9 to 7.5
TSS	111	86 to 322
TDS	1687	214 to 9,958
SO_4 =	813	485 to 3,507
Cd	0.13	<0.01 to 0.50
Cu	0.60	0.18 to 1.6
Fe	90	3.6 to 522
Pb	0.070	0.01 to 0.35
Ag	0.01	<0.005 to 0.20
Zn	44	1.5 to 76
Hg	<0.001	<0.001
As	0.021	<0.01 to 0.040

Based on industry data collected during the period of 1970 through 1978.

*All metals concentrations are based on "total" analyses

**Value in pH units

TABLE VIII-29. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT LEAD/ZINC MINE 3113

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
		MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	53	3.2	3.0 - 3.3	n.a.	n.a.
TSS	7	112	104 - 124	n.a.	n.a.
Sb		<0.1	<0.1	n.a.	n.a.
As		0.013	0.005 - 0.030	n.a.	n.a.
Be		<0.003	<0.003	n.a.	n.a.
Cd		0.23	0.22 - 0.24	0.23	0.23 - 0.24
Cr		0.011	0.010 - 0.015	n.a.	n.a.
Cu		1.5	1.3 - 1.6	1.4	1.4 - 1.5
Pb		0.088	0.033 - 0.12	n.a.	n.a.
Hg		<0.0002	<0.0002	n.a.	n.a.
Ni		0.074	0.060 - 0.090	n.a.	n.a.
Se		0.006	0.003 - 0.010	n.a.	n.a.
Ag		n.a.	n.a.	n.a.	n.a.
T1		<0.05	<0.05	n.a.	n.a.
Zn		71	67 - 74	57	56 - 60
Fe		69	50 - 80	25	22 - 29
$SO_4^=$		1063	925 - 1320	n.a.	n.a.

Based on observations made in period 24 through 28 September 1978.

n.a. = Not Analyzed.

TABLE VIII-30. SUMMARY OF PILOT-SCALE TREATABILITY STUDIES AT MINE 3113

EXPERIMENTAL SYSTEM	UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATIONS* ATTAINED (mg/l)						
		pH†	TSS	Cd	Cu	Pb	Zn	Fe
A	Lime Addition to pH ~ 9.5, Sedimentation	9.3** (9.1 to 9.7)	33	0.025	0.10	<0.02	4.3	4.8
B	Lime Addition to pH ~ 9.5, Sedimentation, Filtration	8.8** (8.4 to 9.0)	<2** (<1 to 3)	0.016** (<0.005 to 0.033	0.02** (0.011 to 0.030)	<0.02** (<0.02)	0.17** (0.095 to 0.30)	0.11** (0.052 to 0.20)
C	Lime Addition to pH ~ 9.5, Aeration, Sedimentation	9.7** (9.7 to 9.8)	35	0.02	0.11	0.02	4.1	4.0
D	Lime Addition to pH ~ 9.5, Aeration, Sedimentation, Filtration	9.5** (9.4 to 9.6)	1	0.005	0.020	<0.02	0.15	0.09
E	Lime Addition to pH ~ 9.5, Polymer Addition, Flocculation, Sedimentation	9.3** (8.8 to 9.8)	10** (10)	0.015** (0.015)	0.05** (0.05)	<0.02** (<0.02 to 0.02)	1.35** (1.3 to 1.4)	1.2** (1.1 to 1.2)
F	Lime Addition to pH ~ 9.5, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	8.9** (8.1 to 9.5)	<1** (<1)	0.009** (0.005 to 0.015)	0.015** (0.01 to 0.02)	0.025** (<0.02 to 0.04)	0.11** (0.075 to 0.16)	0.5** (0.04 to 0.06)
G	Lime Addition to pH ~ 8.5, Aeration, Polymer Addition, Flocculation, Sedimentation	8.4** (7.5 to 8.3)	6	0.02	0.02	0.08	1.9	0.65
H	Lime Addition to pH ~ 8.5, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	7.9** (7.3 to 8.3)	<1** (<1)	0.012** (0.010 to 0.015)	<0.01** (<0.01 to 0.01)	<0.02** (<0.02 to 0.02)	0.15** (0.10 to 0.21)	0.027** (0.02 to 0.04)
I	Lime Addition to pH ~ 10.5, Aeration, Polymer Addition, Flocculation, Sedimentation	10.5** (10.2 to 10.7)	15	0.005	0.02	<0.02	0.67	0.60
J	Lime Addition to pH ~ 10.5, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	10.2** (9.5 to 10.5)	<1** (<1 to 1)	<0.005** (<0.005)	0.013** (0.01 to 0.015)	<0.02** (<0.02)	0.027** (0.02 to 0.03)	0.03** (0.02 to 0.04)

* Metals concentrations are based on "total" analyses

† Value in pH units

** Average concentrations attained

( ) Range of concentrations attained

TABLE VIII-31. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT ALUMINUM MINE 5102

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS (TOTAL/ DISSOLVED)	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
		MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	20	6.5	5.8 - 7.1	n.a.	n.a.
TSS	21	3	<1 - 5	↓	↓
Sb	21	<0.2	<0.2	↓	↓
As	19	<0.0005	<0.0005	↓	↓
Be	21	<0.005	<0.005	↓	↓
Cd	21	0.005	<0.005-0.010	↓	↓
Cr	21	<0.01	<0.01-0.010	↓	↓
Cu	21	<0.01	<0.01-0.015	↓	↓
Pb	21	<0.05	<0.05	↓	↓
Hg	21	<0.0002	<0.0002	↓	↓
Ni	21	<0.03	<0.03-0.040	↓	↓
Se	19	<0.002	<0.002	↓	↓
Ag	21	<0.01	<0.01-0.015	↓	↓
T1	21	<0.03	<0.03	↓	↓
Zn	21/12	0.01	0.005-0.020	≤0.01	<0.01-0.010
Al	21/12	0.49	<0.2-0.7	≤0.02	<0.02-0.020
Fe	21/12	0.15	0.020-0.24	≤0.02	<0.02-0.020
Phenol	5	0.007	<0.002-0.011	n.a.	n.a.

Based on observations made in period 10 through 16 October 1978.

n.a. = Not Analyzed.

TABLE VIII-32. SUMMARY OF PILOT-SCALE TREATABILITY STUDIES AT MINE 5102

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATION* ATTAINED			
	pH†	TSS	Al	Fe
No Treatment (Control)	6.5** (5.8 to 7.1)	3** (<1 to 5)	0.49** (<0.2 to 0.70)	0.15** (0.020 to 0.24)
Lime Addition to pH $\sim$ 8.2, Aeration, Polymer Addition, Flocculation, Sedimentation	8.2	1	<0.2	0.15
Lime Addition to pH $\sim$ 8.2, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	8.0** (7.7 to 8.4)	<1** (< 1)	<0.2** (<0.2)	0.067** (0.06 to 0.07)
Lime Addition to pH $\sim$ 9.0, Aeration, Polymer Addition, Flocculation, Sedimentation	9.0	7	<0.2	0.06
Lime Addition to pH $\sim$ 9,0, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	8.5	<1	0.2	0.08
Lime Addition to pH $\sim$ 10.4, Aeration, Flocculation, Sedimentation	10.4	34	0.2	0.2
Lime Addition to pH $\sim$ 10.2, Aeration, Polymer Addition, Flocculation, Sedimentation	10.2	5	0.3	0.23
Lime Addition to pH $\sim$ 10.2, Aeration, Polymer Addition, Flocculation, Sedimentation, Filtration	10.0** (9.8 to 10.2)	< 1** (<1 to 1)	<0.27** (<0.2 to 0.4)	0.073** (0.07 to 0.08)
Filtration	6.5** (5.8 to 6.8)	< 1** (<1 to 1)	< 0.2** (<0.2 to 0.2)	0.040** (0.2 to 0.10)

* All metals concentrations are based on "total" analyses

† Value in pH units

**Average concentrations attained

( ) Range of concentrations attained

TABLE VIII-33. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT MILL 9402 (ACID LEACH MILL WASTEWATER)

POLLUTANT PARAMETER	PERIOD OF OBSERVATIONS (DATES)	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
			MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	11/3-8/1978	5	1.7	1.6-1.9		--
TSS			40	24-48		
Sb			<0.5	<0.5	--	--
As			2.5	1.6-3.3		--
Be			0.03	0.03	--	
Cd			0.05	0.04-0.05	--	
Cr			0.67	0.46-0.82	0.66	0.44-0.76
Cu			4.0	2.7-6.0	3.7	2.6-5.8
Pb			0.93	0.65-1.1	0.88	0.61-1.0
Hg			<0.0002	<0.0002	--	--
Ni			1.4	1.3-1.6	--	--
Se			2.0	1.1-2.6	--	--
Ag			<0.1	<0.1	--	--
T1			<0.2	<0.2	--	--
Zn			6.1	4.8-7.5	5.8	4.8-7.1
V			100	80-110	97	77-110
Mo			10	9-12	8.9	6.3-11
Fe			1900	1800-2000	1860	1800-2000
Mn			118	100-130	116	95-125
Al			786	640-900	758	640-890
SO_4			21760	13,400-34,300	--	--
TDS		↓	--	--	31,520	28,100-35,300
Ra_{226}		3/4	99.7±1%	(62.6-133)±1%	88.3±1%	(58.4-127)±1%
U	↓	3/4	17	(12±9%) to (20±12%)	16	(12±9%) to (20±12%)

TABLE VIII-34. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER AT MILL 9402 (ACID LEACH MILL WASTEWATER)

POLLUTANT PARAMETER	PERIOD OF OBSERVATIONS (DATES)	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
			MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	12/5/78- 12/11/78	8	1.6	1.4-1.8	--	--
TSS	↓	3	168	142-215	--	--
Cr		3	0.86	0.73-0.98	0.82	0.71-0.93
Cu		3	3.2	2.9-3.6	3.1	2.9-3.4
Pb		3	1.3	1.1-1.5	1.3	1.2-1.3
Ni		3	1.0	0.95-1.1	0.93	0.87-1.0
V		3	109	97-110	93	89-98
Mo		3	17	15-20	17	15-19
Fe		3	3,670	3200-4100	2,331	2,100-2,600
Mn		3	287	260-310	157	150-160
Al		3	1,840	1,640-2,220	1,330	1,170-1,480
SO$_4$		3	19,600	18,400-20,400	--	--
TDS		3	--	-	32,200	29,700-34,200
Ra$_{226}$		3	154.8±1%	(168)±1%	129±1%	(121±1%)-135±1%)
U		3	19.8	16-24.1	20±12%	(16±12%)-(24±12%)

TABLE VIII-35. SUMMARY OF WASTEWATER TREATABILITY RESULTS USING A LIME ADDITION/BARIUM CHLORIDE ADDITION/SETTLE PILOT-SCALE TREATMENT SCHEME

PARAMETER	OPTIMUM CONDITIONS FOR REMOVAL	REMOVAL EFFICIENCY (%)	FINAL CONCENTRATION ATTAINED (mg/l)	
			Total	Dissolved
TDS	pH of 9-9.5	42-78	–	5590-9740
Cu	pH 9.5	91-98	0.11-0.18	0.060-0.15
Pb	pH 8.2-9.5	52-90	< 0.20	< 0.014
Zn	pH 6.8-9.5; Final TSS < 40	98 to > 99	0.10	< 0.020-0.030
Ni	pH 5.8-9.5	90-96	< 0.040	< 0.040
Cr	pH 5.8-9.5	87-96	< 0.050	< 0.040
Fe	pH 8.2-9.5	98 to > 99	0.80-32	0.10-1.0
Mn	pH 8.2-9.5	92 to > 99	0.88-4.9	0.43-4.2
Al	pH 5.8-9.5	96 to > 99	0.90-17	0.50-5.0
V	pH 5.8-9.5	97 to > 99	0.20-1.3	≤ 0.20
Mo	pH 5.8-6.1	73	4.6	2.2
Ra 226	$BaCl_2$ dosage of 51-63 mg/l; Final TSS < 200	96-97	3.9-4.0*	1.0-2.3*
U	Final TSS < 50	78-97	0.30-2.5	0.20-0.50
NH_3	None attained	0-25	123-292	–

*Values in picocuries per liter (pc/l)

TABLE VIII-36. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT MILL 9401

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
		MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	20	10.0	9.9 - 10.1	–	–
TSS	6	945	156 - 1528	–	–
Sb	6	<0.50	<0.50	–	–
As	5/4	4.6	4.0 - 5.0	4.25	4.0 - 5.0
Be	6	<0.010	<0.010	–	–
Cd	6	<0.020	<0.020	<0.020	<0.020
Cr	6	<0.050	<0.050	–	–
Cu	6	0.060	0.040 - 0.080	–	–
Pb	6	<0.10	<0.10	–	–
Hg	5	0.0002	0.0002	–	–
Ni	6	<0.10	<0.10	–	–
Se	5/4	19	17 - 20	19	16 - 20
Ag	6	<0.10	<0.10	–	–
T1	6	<0.10	<0.10	–	–
Zn	6	0.023	<0.020 - 0.030	<0.020	<0.020
Mo	6	106	95 - 110	108	100 - 110
V	5	26	24 - 27	27	25 - 27
U	5/3	58.6	55 - 63	39	8 - 57
Ra 226**	5	163	30-677	29	18 - 48

* pH units

**pCi/l

TABLE VIII-37. SUMMARY OF TREATED EFFLUENT QUALITY ATTAINED WITH PILOT SCALE TREATMENT SYSTEM AT MILL 9401

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATIONS ATTAINED (mg/l)*									
	pH**	TSS	TDS	$SO_4^=$	AS	SE	Mo	V	U	Ra 226†
Fe (SO_4) (1000 mg/l), Aeration, Non-Ionic Polymer, $BaCl_2$ (60 mg/l), Flocculation, Sedimentation, Filtration	10.0	1744	21700	9380	<5.0 (d)	9.6 (d)	90	8.0	59 ±13%	3.1 ±2%
IX, Fe (SO_4) (330 mg/l), Aeration, Non-Ionic Polymer, $BaCl_2$ (15 mg/l), Flocculation, Sedimentation, Filtration	9.0	2.0	24300	10500	<5.0 (d)	8.5 (d)	55	9.0	6.0 ±9%	2.5 ±6%
IX, Fe(SO_4) (500 mg/l), Aeration, Non-Ionic Polymer, $BaCl_2$ (120 mg/l), Flocculation, Sedimentation, Filtration	8.0	468	23200	9830	<5.0 (d)	8.5 (d)	55	5.0	0.72 ±18%	44 ±7.3%

* Metals are total metals unless otherwise indicated by a (d) = dissolved

**pH units

† pC1/l

TABLE VIII-38. RESULTS OF BENCH-SCALE ACID/ALKALINE MILL WASTEWATER NEUTRALIZATION

ACID/ALKALINE MILL WASTEWATER MIX RATIO BY VOLUME	THEORETICAL PARAMETER CONCENTRATION (mg/l)								MEASURED PARAMETER CONCENTRATION (mg/l)									APPARENT REMOVAL (percent)							
	Fe		Mo		Se		V		Fe		Mo		Se		V		pH	Fe		Mo		Se		V	
	T	D	T	D	T	D	T	D	T	D	T	D	T	D	T	D		T	D	T	D	T	D	T	D
500/0	1900	1860	10	8.9	1.9	–	100	97	1900	1860	10	8.9	1.94	–	100	97	1.7	0	0	0	0	0	–	0	0
500/300	1190	1160	46	46	8.3	–	72	71	520	520	1.3	0.70	10	–	1.5	<0.50	3.9	56.0	55.3	97.2	98.5	0	–	97.9	>99.3
500/400	1060	1030	53	53	9.5	–	67	66	480	480	1.5	2.0	12	–	1.0	1.7	4.3	54.5	53.5	97.2	96.2	0	–	98.5	97.4
400/500	850	830	63	64	11	–	59	58	400	350	8.9	6.7	15	–	4.5	<0.5	4.9	52.4	57.7	85.9	89.5	0	–	92.4	99.1
300/500	710	700	70	71	13	–	54	53	283	270	30	30	–	–	1.8	2.0	5.8	60.3	61.3	57.1	57.6	0	–	96.7	96.2
200/500	540	530	79	80	14	–	47	47	150	140	59	56	18	–	2.5	0.5	6.6	72.6	73.7	24.9	29.7	0	–	94.7	98.9
150/500	440	430	84	85	15	–	43	43	93	37	59	60	20	–	6.0	1.0	8.2	78.8	91.4	29.6	29.5	0	–	86.1	97.7
0/500	<0.20	<0.20	110	110	19	19	26	26	<0.20	<0.20	110	110	19	19	26	27	10.0	0	0	0	0	0	–	0	0

† $\frac{\text{(theoretical concentration)} - \text{(measured concentration)}}{\text{theoretical concentration}}$ (100)

TABLE VIII-39. CHARACTERIZATION OF INFLUENT TO WASTEWATER TREATMENT PLANT AT COPPER MINE/MILL/SMELTER/REFINERY 2122 (SEPTEMBER 5-7, 1979)

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	TOTAL CONCENTRATION (mg/l)		DISSOLVED CONCENTRATION (mg/l)	
		MEAN	RANGE	MEAN	RANGE
pH	3	2.46	2.4-2.55	–	–
TSS	3	297	26-790	–	–
Fe	3	17	13-26	11.1	9.4-13
Sb	3	0.015	0.015-0.016	0.014	0.013-0.016
As	3	5.16	4.8-5.40	5.10	4.30-5.80
Be	3	<0.0005	<0.0005	<0.0005	<0.0005
Cd	3	0.092	0.076-0.120	0.102	0.081-0.140
Cr	3	0.146	0.098-0.190	0.134	0.094-0.160
Cu	3	9.43	8.40-10.0	3.76	3.60-3.90
Pb	3	5.56	5.0-6.10	3.66	3.30-4.0
Hg	3	0.014	0.010-0.017	0.002	0.002-0.003
Ni	3	0.160	0.130-0.190	0.17	0.16-0.18
Se	3	0.036	0.034-0.037	0.050	0.035-0.077
Ag	3	0.028	0.016-0.039	<0.021	<0.01-0.027
Tl	3	<0.004	<0.002-0.006	0.005	0.004-0.006
Zn	3	1.73	1.50-2.10	1.8	1.6-2.10
CN	3	<0.03	<0.02-0.05	–	–
Tot. Phenolics	3	0.14	0.04-0.3	–	–

TABLE VIII-40. CHARACTERIZATION OF EFFLUENT FROM WASTEWATER TREATMENT PLANT (INFLUENT TO PILOT-SCALE TREATMENT PLANT) AT COPPER MINE/MILL/SMELTER/REFINERY 2122 (SEPTEMBER 5-10, 1979)

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	TOTAL CONCENTRATION (mg/l)		DISSOLVED CONCENTRATION (mg/l)	
		MEAN	RANGE	MEAN	RANGE
pH	14	8.2	7.2-8.65	–	–
TSS	14	14.7	3.0-61.0	–	–
Fe	14	0.66	0.06-1.1	0.04	0.03-0.06
Sb	14	0.014	0.009-0.032	<0.013	<0.005-0.023
As	14	1.95	1.0-4.0	1.6	0.9-2.4
Be	14	<0.0005	<0.0005	<0.0005	<0.0005
Cd	14	0.044	0.028-0.065	0.039	0.021-0.070
Cr	14	0.054	0.035-0.077	0.047	0.029-0.067
Cu	14	0.374	0.120-0.650	0.079	0.018-0.210
Pb	14	0.253	0.005-0.410	<0.003	<0.002-0.005
Hg	14	0.002	<0.001-0.005	<0.002	<0.001-0.003
Ni	14	0.146	0.110-0.190	0.143	0.099-0.200
Se	14	0.110	0.021-0.230	0.100	0.020-0.180
Ag	14	<0.020	<0.01-0.036	<0.019	<0.01-0.033
Tl	14	0.004	<0.002-0.007	<0.005	<0.002-0.008
Zn	14	0.309	0.120-0.730	0.154	0.018-0.660
CN	14	<0.02	<0.02	–	–
Tot. Phenolics	14	0.41	0.02-5.2	–	–

TABLE VIII-41. SUMMARY OF TREATED EFFLUENT QUALITY FROM PILOT-SCALE TREATMENT PROCESSES AT COPPER MINE/MILL/SMELTER/REFINERY 2122 (SEPTEMBER 5-10, 1979)

TEST NUMBER	TREATMENT APPLIED	EFFLUENT CONCENTRATION (TOTAL, mg/l)					
		pH*	TSS	Cu	Pb	Zn	Fe
01	Filtration† 0.15 $m^3/min/m^2$ (3.6 gpm/ft^2)	7.2	4	0.084	0.004	0.310	0.09
02	Filtration† 0.26 $m^3/min/m^2$ (6.3 gpm/ft^2)	7.6	3	0.095	0.015	0.260	0.08
03	Filtration† 0.37 $m^3/min/m^2$ (9.1 gpm/ft^2)	8.1	2	0.094	0.010	0.120	0.06
04	Filtration† 0.43 $m^3/min/m^2$ (10.6 gpm/ft^2)	8.1	2	0.093	0.015	0.140	0.07
05	Filtration† 0.50 $m^3/min/m^2$ (12.2 gpm/ft^2)	8.3	2	0.100	0.009	0.210	0.06
06	One hour settling test	8.5	8	0.250	0.043	0.170	0.54
07	Settling -45 min Filtration† with Lime Addition to pH 9.1 0.38 $m^3/min/m^2$ (9.4 gpm/ft^2)	8.95	<1	0.060	0.004	0.031	0.04
08	Filtration† with Lime Addition to pH 9.0 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	8.95	1	0.070	0.005	0.043	0.04
09	Filtration† with Lime Addition to pH 9.5 0.37 $m^3/min/m^2$ (9.5 gpm/ft^2)	9.05	1	0.064	0.017	0.023	0.07

†Dual media filtration (anthrafilt and silica sand)

*Field pH values

TABLE VIII-41. SUMMARY OF TREATED EFFLUENT QUALITY FROM PILOT-SCALE TREATMENT PROCESSES AT COPPER MINE/MILL/SMELTER/REFINERY 2122 (SEPTEMBER 5-10, 1979) (Continued)

TEST NUMBER	TREATMENT APPLIED	EFFLUENT CONCENTRATION (TOTAL, mg/l)					
		pH*	TSS	Cu	Pb	Zn	Fe
10	Filtration† 5 mins 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	–	1	0.058	0.011	0.038	0.06
11	Filtration† 6 hours 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	8.45	1	0.073	0.002	0.110	0.07
12	Filtration† 12 hours 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	8.3	<1	0.044	0.018	0.097	0.04
13	Filtration† 18 hours 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	8.6	1	0.110	0.011	0.038	0.03
16	Filtration† with Lime Addition to pH 10.0 0.37 $m^3/min/m^2$ (9.1 gpm/ft^2)	9.75	2	0.056	<0.002	0.110	0.03

†Dual media filtration (anthrafilt and silica sand)

*Field pH values

TABLE VIII-42. CHARACTERIZATION OF EFFLUENT FROM WASTEWATER TREATMENT SYSTEM (INFLUENT TO PILOT-SCALE TREATMENT PLANT) AT COPPER MINE/MILL/SMELTER/REFINERY 2121 (SEPTEMBER 18-19, 1979)*

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	TOTAL CONCENTRATION (mg/l)		DISSOLVED CONCENTRATION (mg/l)	
		MEAN	RANGE	MEAN	RANGE
pH**	3	7.5	7.4-7.65	–	–
TSS	3	4.1	3.9-4.2	–	–
Fe	3	0.27	0.25-0.29	0.088	0.071-0.110
Sb	3	<0.003	<0.003	<0.003	<0.003
As	3	<0.002	<0.002	<0.006	<0.002-0.015
Be	3	<0.0005	<0.0005	<0.0005	<0.0005
Cd	3	<0.008	<0.005-0.013	<0.008	<0.005-0.013
Cr	3	<0.008	<0.005-0.015	<0.008	<0.005-0.013
Cu	3	0.023	0.022-0.025	0.022	0.021-0.023
Pb	3	0.004	0.003-0.006	0.006	0.005-0.009
Hg	3	<0.001	<0.001	<0.001	<0.001
Ni	3	0.054	0.053-0.055	0.055	0.053-0.057
Se	3	<0.005	<0.005	<0.005	<0.005
Ag	3	<0.01	<0.01	<0.01	<0.01
Tl	3	<0.003	<0.003	<0.003	<0.003
Zn	3	0.015	0.011-0.019	0.032	0.018-0.049
Tot. Phenolics	2	0.01	0.007-0.013	–	–

* Grab samples

**pH units

TABLE VIII-43. SUMMARY OF TREATED EFFLUENT QUALITY FROM PILOT-SCALE TREATMENT PROCESSES AT COPPER MINE/MILL/SMELTER/REFINERY 2121 (SEPTEMBER 18-19, 1979)

TEST NUMBER	TREATMENT APPLIED	EFFLUENT CONCENTRATION (TOTAL, mg/l)					
		pH*	TSS	Cu	Pb	Zn	Fe
01	Filtration† 0.26 $m^3/min/m^2$ (6.5 gpm/ft^2)	7.2	1.1	0.038	0.005	0.024	0.21
02	Filtration† 0.38 $m^3/min/m^2$ (9.3 gpm/ft^2)	7.3	1.6	0.020	0.004	0.011	0.11
03	Filtration with Lime Addition to pH 8.8 0.37 $m^3/min/m^2$ (9.1 gpm/ft^2)	7.7	1.1	0.020	0.004	0.016	0.17

†Dual media filtration (anthrafilt and silica sand)

*Field pH values

TABLE VIII-44. HISTORICAL DATA SUMMARY FOR IRON ORE MINE/MILL 1108 (FINAL DISCHARGE)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN	STANDARD DEVIATION	RANGE
pH	Jan. 74 - Apr. 77	1/ mo. (mo. ave.)	35	7.1 (ave. of mo. means)	0.2	6.7 – 7.7
pH	Jan. 74 - Apr. 77	3 - 5/mo	148	7.1 (daily ave)	0.3	6.5 – 8.0
TSS	Jan. 74 - Apr. 77	1/ mo. (mo. ave.)	35	5.9 (ave. of mo. means)	4.8	2 – 28
TSS	Jan. 74 - Apr. 77	4 - 31/mo.	804	6.1 (daily ave.)	6.8	1.0 – 70
Fe (dissolved)	Jan. 74 - Apr. 77	3 - 5/mo. (mo. ave.)	35	0.36 (ave. of mo. means)	0.47	0.05 – 1.92
Fe (dissolved)	Jan. 74 - Apr. 77	3 - 5/mo	147	0.35 (daily ave.)	0.58	0.01 – 3.6

TABLE VIII-45. HISTORICAL DATA SUMMARY FOR COPPER/SILVER MINE/MILL/SMELTER/REFINERY 2121 (FINAL TAILINGS POND DISCHARGE)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN***	STANDARD DEVIATION	RANGE†
Flow (m^3/day)	March 1975 - Dec. 1979	Continuous	–	79,107	36,714	21,953 - 168,400
pH (Std. Units)	March 1975 - Dec. 1979	1/month††	58	7.8	0.4	7.2 - 9.0
TSS (mg/l)**	March 1975 - Dec. 1979	1/month††	58	5.5	2.9	2 - 29
Copper (mg/l)	March 1975 - Dec. 1979	1/month	59	<0.023	0.015	0.003 - 0.065
Zinc (mg/l)	March 1975 - Dec. 1979	1/month	59	<0.026	0.019	<0.002 - 0.09
Chlorides (mg/l)	March 1975 - Dec. 1979	1/month	57	866	262	354 - 1,494

* All metals expressed as total metals unless otherwise specified.

** TSS values measured by turbidity and converted to mg/l.

† Range of concentrations based on maximum and minimum data between March 1975 and April 1977, and average concentrations between May 1977 and December 1979.

†† Monthly averages based on almost continuous daily monitoring.

***Mean of monthly averages.

TABLE VIII-46. HISTORICAL DATA SUMMARY FOR COPPER MINE/MILL 2120 (TAILING-POND OVERFLOW: TREATMENT OF UNDERGROUND MINE, MILL, AND LEACH-CIRCUIT WASTEWATER STREAMS)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Sept. 75 - June 77	15 - 31/mo.	538	9.53 (daily ave.)	1.02	6.4 – 12.0
pH	Sept. 75 - June 77	1/mo. (mo. ave.)	22	9.23 (ave. of mo. means)	1.07	7.2 – 11.0
TSS	Sept. 75 - June 77	15 - 31/mo.	536	9.6 (daily ave.)	10.6	1 – 132
TSS	Sept. 75 - June 77	1/mo. (mo. ave.)	22	10 (ave. of mo. means)	3	5 – 19
Cu	Sept. 75 - June 77	15 - 31/mo.	536	0.065 (daily ave.)	0.081	0.01 – 0.88
Cu	Sept. 75 - June 77	1/mo. (mo. ave.)	22	0.07 (ave. of mo. means)	0.05	0.02 – 0.27
Pb	Sept. 75 - June 77	1 - 5/mo.	82	< 0.01 (daily ave.)	0	none
Pb	Sept. 75 - June 77	1/mo. (mo. ave.)	22	< 0.01 (ave. of mo. means)	0	none
Zn	Sept. 75 - June 77	15 - 31/mo.	535	0.177 (daily ave.)	0.556	0.01 – 7 1
Zn	Sept. 75 - June 77	1/mo. (mo. ave.)	22	0.23 (ave. of mo. means)	0.43	0.02 – 1.71

*All metals expressed as total metals unless otherwise specified.

TABLE VIII-47. HISTORICAL DATA SUMMARY FOR COPPER MINE/MILL 2120 (TREATMENT SYSTEM – BARREL POND – EFFLUENT: TREATMENT OF MILL WATER AND OPEN-PIT MINE WATER

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 75 - Sept. 77	25-31/mo.	953	10.8 (daily ave.)	1.1	2.9 – 13.1
pH	Jan. 75 - Sept. 77	1/mo.	33	10.7 (ave. of mo. means)	0.6	9.7 – 12.4
TSS	Jan. 75 - Sept. 77	25-31/mo.	954	12 (daily ave.)	10	0 – 120
TSS	Jan. 75 - Sept. 77	1/mo.	33	12 (ave. of mo. means)	4	7 – 22
Cu	Jan. 75 - Sept. 77	25-31/mo.	957	0.09 (daily ave.)	0.09	< 0.01 – 0.98
Cu	Jan. 75 - Sept. 77	1/mo.	33	0.10 (ave. of mo. means)	0.05	0.04 – 0.27
Pb	Jan. 75 - Sept. 77	3-5/mo.	33	0.015 (ave. of mo. means)	0.012	< 0.01 – 0.06
Hg	Apr. 75 - Sept. 77	3-5/mo.	30	0.0003 (ave. of mo. means)	0.0001	< 0.00005 – 0.0006
Zn	Jan. 75 - Sept. 77	25-31/mo.	957	0.14 (daily ave.)	0.16	< 0.01 – 2.00
Zn	Jan. 75 - Sept. 77	1/mo.	33	0.15 (ave. of mo. means)	0.09	0.04 – 0.36

*All metals expressed as total metals unless otherwise specified.

TABLE VIII-48. HISTORICAL DATA SUMMARY FOR LEAD MINE/MILL 3105 (TREATED MINE EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	49	8.1 (ave. of mo. means)	0.23	7.4 – 8.5
TSS	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	43	3.0 (ave. of mo. means)	2.0	1 – 9
CN	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	49	< 0.02 (ave. of mo. means)	0.0	na
Cu	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	49	0.006 (ave. of mo. means)	0.001	< 0.005 – 0.010
Pb	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	49	0.043 (ave. of mo. means)	0.023	0.01 – 0.12
Zn	Jan. 74 - Jan. 78	1/mo. (mo. ave.)	49	0.026 (ave. of mo. means)	0.022	0.005 – 0.11

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-49. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE 3130 (UNTREATED MINEWATER)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Aug. 77	unk	unk	7.7	n.a.	7.2 - 8.3
TSS	Aug. 77 - Oct. 77	6/mo.	18	42.6	20.2	5.6 - 90.2
TSS	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	44.2 (ave. of mo. means)	na	33.2 - 61.9
CN	Aug. 77 - Oct. 77	1/mo.	3	0.11	na	0.04 - 0.16
CN	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	0.11 (ave. of mo. means)	na	0.04 - 0.16
Pb	July 77 - Oct. 77	6/mo.	20	0.93	0.357	0.30 - 1.60
Pb	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	1.00 (ave. of mo. means)	na	0.70 - 1.40
Zn	July 77 - Oct. 77	6/mo.	20	1.25	0.52	0.45 - 2.50
Zn	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	1.35 (ave. of mo. means)	na	1.09 - 1.88

*All metals expressed as total metals unless otherwise specified.
unk = unknown
na = not applicable

TABLE VIII-50. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE 3130 (TREATED EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	June 77 - Sept. 77	unk	unk	8.8	na	8.5 – 8.9
pH	June 77 - Sept. 77	unk	4	8.8 (ave. of mo. means)	na	8.6 – 8.9
TSS	Aug. 77 - Oct. 77	8/mo.	23	2.12	1.34	< 0.1 – 6.1
TSS	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	2.1 (ave. of mo. means)	na	1.8 – 2.4
Cu	Aug. 77 - Oct. 77	1/mo.	3	0.12	na	0.05 – 0.25
Cu	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	0.12 (ave. of mo. means)	na	0.05 – 0.25
CN	June 77 - Oct. 77	1/mo.	5	0.064	na	0.022 – 0.150
CN	June 77 - Oct. 77	1/mo. (mo. ave.)	5	0.064 (ave. of mo. means)	na	0.022 – 0.150
Pb	July 77 - Oct. 77	8/mo	25	0.079	0.021	< 0.05 – 0.13
Pb	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	0.08 (ave. of mo. means)	na	0.067 – 0.097
Zn	July 77 - Oct. 77	8/mo.	25	0.32	0.085	0.18 – 0.57
Zn	Aug. 77 - Oct. 77	1/mo. (mo. ave.)	3	0.33 (ave. of mo. means)	na	0.24 – 0.39

*All metals expressed as total metals unless otherwise specified.
unk = unknown
na = not applicable

TABLE VIII-51. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3101 (TAILING-POND DECANT TO POLISHING PONDS)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Oct. 75 - Aug. 77	4 - 5/mo.	197	9.3 (daily ave.)	1.5	7.0 - 11.3
pH	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	9.2 (ave. of mo. means)	1.3	6.7 - 11.2
Cu	Oct. 75 - Aug. 77	4 - 5/mo.	196	0.054 (daily ave.)	0.072	0.002 - 0.490
Cu	Jan 74 - Aug. 77	1/mo. (mo. ave.)	44	0.055 (ave. of mo. means)	0.060	0.008 - 0.323
Pb	Oct. 75 - Aug. 77	4 - 5/mo.	196	0.025 (daily ave.)	0.049	0.004 - 0.525
Pb	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	0.040 (ave. of mo. means)	0.049	0.004 - 0.303
Zn	Oct. 75 - Aug. 77	4 - 5/mo.	197	0.193 (daily ave.)	0.264	0.010 - 2.20
Zn	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	0.294 (ave. of mo. means)	0.401	0.038 - 2.15

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-52. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3101 (FINAL DISCHARGE FROM POLISHING POND)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Oct. 75 - Aug. 77	4 - 5/mo.	101	8.1 (daily ave.)	1.3	6.6 – 11.2
pH	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	8.3 (ave. of mo. means)	1.3	6.9 – 10.9
TSS	Oct. 75 - Aug. 77	4 - 5/mo.	14	8.4 (daily ave.)	na	0.3 – 26.0
Cu	Oct. 75 - Aug. 77	4 - 5/mo.	100	0.024 (daily ave.)	0.027	0.002 – 0.124
Cu	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	0.020 (ave. of mo. means)	0.017	0.006 – 0.076
Pb	Oct. 75 - Aug. 77	4 - 5/mo.	98	0.009 (daily ave.)	0.008	0.004 – 0.052
Pb	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	0.020 (ave. of mo. means)	0.018	0.005 – 0.082
Zn	Oct. 75 - Aug. 77	4 - 5/mo.	101	0.211 (daily ave.)	0.123	0.026 – 0.548
Zn	Jan. 74 - Aug. 77	1/mo. (mo. ave.)	44	0.176 (ave. of mo. means)	0.096	0.028 – 0.390

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-53. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3102 (FINAL DISCHARGE)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	7.9 (ave. of mo. means)	0.3	7.6 – 8.4
TSS	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	2.6 (ave. of mo. means)	3.8	0 – 10
CN	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	< 0.02 (ave. of mo. means)	0	na
Cu	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	0.001 (ave. of mo. means)	0.001	$< 0.001 - 0.003$
Pb	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	0.002 (ave. of mo. means)	0.002	$< 0.001 - 0.007$
Zn	Dec. 73 - Sept. 74	1/mo. (mo. ave.)	10	0.01 (ave. of mo. means)	0.02	$< 0.001 - 0.07$

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-54. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3103 (TAILING-POND EFFLUENT TO SECOND SETTLING POND)

PARAMETER*	MONITORING PERIOD†	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Feb. 74 - Nov. 77	1/mo.	32**	7.84	0.26	7.4 - 8.5
TSS	Feb. 74 - Nov. 77	1/mo.	35	<1.63	0.8	<1.0 - 4.0
Cu	Feb. 74 - Nov. 77	1/mo.	40	0.023	0.013	0.005 - 0.058
Pb	Feb. 74 - Nov. 77	1/mo.	40	0.154	0.0685	0.038 - 0.33
Zn	Feb. 74 - Nov. 77	1/mo.	40	0.309	0.174	0.030 - 0.79

*All metals expressed as total metals unless otherwise specified.

†The following months were excluded due to missing data: Oct. 1975, June 1975, Sept. 1975, Oct. 1976, and Jan. 1977.

**Several illegible data points excluded.

TABLE VIII-55. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3103 (EFFLUENT FROM SECOND SETTLING POND)

PARAMETER*	MONITORING PERIOD†	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Feb. 74 - Nov. 77	1/mo.	33**	7.93	0.22	7.5 – 8.4
TSS	Feb. 74 - Nov. 77	1/mo.	38	< 1.55	1.48	1.0 – 8.0
Cu	Feb. 74 - Nov. 77	1/mo.	39	0.013	0.007	0.002 – 0.034
Pb	Feb. 74 - Nov. 77	1/mo.	40	0.058	0.028	0.01 – 0.122
Zn	Feb. 74 - Nov. 77	1/mo.	40	0.110	0.086	0.018 – 0.440

*All metals expressed as total metals unless otherwise specified.

†The following months were excluded due to missing data: Oct. 1974, June 1975, July 1975, Sept. 1975, Oct. 1975, and Jan. 1977

**Several illegible data points excluded.

TABLE VIII-55. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3103 (EFFLUENT FROM SECOND SETTLING POND)

PARAMETER*	MONITORING PERIOD †	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Feb. 74 - Nov. 77	1/mo.	33**	7.93	0.22	7.5 – 8.4
TSS	Feb. 74 - Nov. 77	1/mo.	38	< 1.55	1.48	1.0 – 8.0
Cu	Feb. 74 - Nov. 77	1/mo.	39	0.013	0.007	0.002 – 0.034
Pb	Feb. 74 - Nov. 77	1/mo.	40	0.058	0.028	0.01 – 0.122
Zn	Feb. 74 - Nov. 77	1/mo.	40	0.110	0.086	0.018 – 0.440

*All metals expressed as total metals unless otherwise specified.

†The following months were excluded due to missing data: Oct. 1974, June 1975, July 1975, Sept. 1975, Oct. 1975, and Jan. 1977

**Several illegible data points excluded.

TABLE VIII-56. HISTORICAL DATA SUMMARY FOR LEAD/ZINC MINE/MILL 3104 (TAILING-LAGOON OVERFLOW)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 74 - Dec. 77	4-5/mo. (mo. ave.)	48	7.8	0.8	6.8 – 10
TSS	Jan. 74 - Dec. 77	4-5/mo. (mo. ave.)	48	6.8	2.9	2.9 – 16
CN	Apr. 77 - Dec. 77	1/mo.	9	<0.1	0	na
Cu	Jan. 74 - Dec. 77	4-5/mo. (mo. ave.)	48	0.06	0.04	0.01 – 0.14
Pb	Jan. 74 - Dec. 77	4-5/mo. (mo. ave.)	48	0.07	0.02	0.03 – 0.12
Zn	Jan. 74 - Dec. 77	4-5/mo. (mo. ave.)	48	0.13	0.07	0.03 – 0.42

*All metals expressed as total metals unless otherwise specified.

na = not applicable

TABLE VIII-57. HISTORICAL DATA SUMMARY FOR LEAD/ZING MILL 3110 (TAILING-LAGOON OVERFLOW)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 74 - Apr. 77	1/mo.	40	7.2	0.4	6.3 – 8.3
TSS	Jan. 74 - Apr. 77	1/mo.	40	5.5	5.1	0.4 – 20
Cu	Jan. 74 - Apr. 77	1/mo.	40	0.02	0.02	0.001 – 0.111
Pb	Jan. 74 - Apr. 77	1/mo.	40	0.05	0.02	0.001 – 0.106
Zn	Jan. 74 - Apr. 77	1/mo.	40	0.20	0.18	0.01 – 0.35

*All metals expressed as total metals unless otherwise specified.

TABLE VIII-58. HISTORICAL DATA SUMMARY FOR MOLYBDENUM MINE 6103 (TREATED EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	July 76 - June 77	1/mo.	12	8.5	0.24	8.1 – 8.9
TSS	July 76 - June 77	1/mo.	12	8.3	5.9	< 5 – 25
As	Apr. 77 - June 77	1/mo.	3	< 0.01	–	none
Cd	July 76 - June 77	1/mo.	12	< 0.01	–	none
Cu	July 76 - June 77	1/mo.	12	< 0.05	–	none
Pb	Apr. 77 - June 77	1/mo.	3	< 0.01	–	none
Mo	July 76 - June 77	1/mo.	12	0.50	0.21	0.21 – 1.0
Zn	July 76 - June 77	1/mo.	12	0.24	0.069	0.15 – 0.4

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-59. HISTORICAL DATA SUMMARY FOR MOLYBDENUM MINE 6101 (TREATED EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 75 - Dec. 76	0-5/mo.	47	7.57 (daily ave.)	0.38	6.5 – 9.0
TSS	Jan. 75 - Dec. 76	0-5/mo.	50	7.6 (daily ave.)	7.5	1 – 34
COD	Jan. 75 - Dec. 76	0-5/mo	33	28.5 (daily ave.)	12	8 – 52
Cd	Jan. 75 - Dec. 76	0-5/mo	44	< 0.02 (daily ave.)	na	< 0.01 – < 0.02
Cu	1972	0-5/mo.	4	< 0.02 (daily ave.)	0.01	< 0.02 – 0.03
CN	Jan. 75 - Dec. 76	0-5/mo.	51	< 0.02 (daily ave.)	0.021	< 0.02 – 0.083
Mo	Jan. 75 - Dec. 76	0-5/mo	43	1.84 (daily ave.)	0.38	1.1 – 2.9
Se	1972	0-5/mo.	4	< 0.005 (daily ave.)	na	none
Zn	Jan. 75 - Dec. 76	0-5/mo.	48	0.077 (daily ave.)	0.18	< 0.01 – 0.90

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-60. HISTORICAL DATA SUMMARY FOR MOLYBDENUM MINE/MILL 6102 (CLEAR POND BLEED STREAM – INFLUENT TO TREATMENT SYSTEM)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
TSS	July - Oct. 1978	1-5/week	33	65.4	180	10 - 1070
Cd	July - Oct. 1978	1-5/week	34	0.02	0.022	<0.01 - 0.15
Cu	July - Oct. 1978	1-5/week	35	0.055	0.016	<0.05 - 0.10
Fe	July - Oct. 1978	1-5/week	33	2.2	2.4	0.60 - 13.0
Mn	July - Oct. 1978	1-5/week	35	7.3	1.2	5.4 - 12.2
Mo	July - Oct. 1978	1-5/week	35	5.6	1.3	3.0 - 8.3
Pb	July - Oct. 1978	1-5/week	35	0.01	0.002	<0.01 - 0.02
Zn	July - Oct. 1978	1-5/week	35	0.84	0.68	0.20 - 2.7
Cyanide	July - Oct. 1978	1-5/week	35	0.06	0.06	<0.01 - 0.2

*All metals expressed as total metals unless otherwise specified

TABLE VIII-61. HISTORICAL DATA SUMMARY FOR MOLYBDENUM MINE/MILL 6102 (FINAL DISCHARGE FROM RETENTION POND – TREATED EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
TSS	July - Oct. 1978	1-5/week	36	9	11.4	<5 - 60
Cd	July - Oct. 1978	1-5/week	37	0.01	0.006	<0.01 - 0.03
Cu	July - Oct. 1978	1-5/week	37	<0.05	0	no range
Fe	July - Oct. 1978	1-5/week	37	0.4	0.32	0.15 - 2.0
Mn	July - Oct. 1978	1-5/week	36	0.26	0.17	0.1 - 0.7
Mo	July - Oct. 1978	1-5/week	37	2.5	1.8	0.5 - 7.0
Pb	July - Oct. 1978	1-5/week	37	<0.01	0	no range
Zn	July - Oct. 1978	1-5/week	37	0.29	0.16	<0.05 - 0.8
Cyanide	July - Oct. 1978	1-5/week	37	<0.01	0	no range

*All metals expressed as total metals unless otherwise specified.

TABLE VIII-62. HISTORICAL DATA SUMMARY; BAUXITE MINE 5102; MINE DRAINAGE; DISCHARGE 008; JANUARY 1979 - DECEMBER 1980

PARAMETER**	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS***	MEAN†	STANDARD DEVIATION	RANGE
Flow (m^3/day)*	Jan. 1979 - Dec. 1980	Continuous	–	10,860	7,470	0 - 38,153
pH (Std. Units)	Jan. 1979 - Dec. 1980	1/week	21	–	–	4.5 - 8.8
TSS (mg/l)††	Jan. 1979 - Dec. 1980	1/week	21	3.7	2.5	0.5 - 13.3
Iron (mg/l)††	Jan. 1979 - Dec. 1980	1/week	21	0.52	0.46	0.08 - 2.61
Aluminum (mg/l)††	Jan. 1979 - Dec. 1980	1/week	21	1.04	0.50	0.22 - 4.02

* Mean flow and standard deviation include zero discharge months. No discharge during July, August and September 1980.

** All metals expressed as total metals unless otherwise specified.

***Only one sample collected during September 1979, December 1979, June 1980 and October 1980.

† Mean of monthly averages.

†† Mean, standard deviation and range for discharging months only. Number rounded to significant figures.

TABLE VIII-63. HISTORICAL DATA SUMMARY; BAUXITE MINE 5102; MINE DRAINAGE; DISCHARGE 009; FEBRUARY 1979 - DECEMBER 1980

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS**	MEAN†	STANDARD DEVIATION	RANGE
Flow (m^3/day)	Feb. 1979 - Dec. 1980	Continuous	–	14,120	8,930	5,791 - 43,868
pH (Std. Units)	Feb. 1979 - Dec. 1980	1/week	21	–	–	6.2 - 8.8
TSS (mg/l)††	Feb. 1979 - Dec. 1980	1/week	21	2.2	0.92	0.5 - 11.2
Iron (mg/l)††	Feb. 1979 - Dec. 1980	1/week	21	0.19	0.08	0.02 - 0.6
Aluminum (mg/l)††	Feb. 1979 - Dec. 1980	1/week	21	0.67	0.37	0.12 - 2.25

* All metals expressed as total metals unless otherwise specified

** No data for April 1980 and July 1980

† Mean of monthly averages

†† Numbers rounded to significant figures

TABLE VIII-64. HISTORICAL DATA SUMMARY: BAUXITE MINE 5102; MINE DRAINAGE; DISCHARGE 010; FEBRUARY 1979 - DECEMBER 1980

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS**	MEAN†	STANDARD DEVIATION	RANGE
Flow (m^3/day)††	Feb. 1979 - Dec. 1980	Continuous	–	7,040	4,845	0 - 17,714
pH (Std. Units)	Feb. 1979 - Dec. 1980	1/week	19	–	–	4.4 - 8.8
TSS (mg/l)***	Feb. 1979 - Dec. 1980	1/week	19	4.4	1.6	0.8 - 14.4
Iron (mg/l***	Feb. 1979 - Dec. 1980	1/week	19	0.25	0.09	0.02 - 0.78
Aluminum (mg/l)***	Feb. 1979 - Dec. 1980	1/week	19	1.47	0.69	0.48 - 5.0

* All metals expressed as total metals unless otherwise specified

** Only one sample collected during February and March 1980

***Mean, standard deviation and range for discharging months only. Numbers rounded to significant figures

† Mean of monthly averages

†† Mean flow and standard deviation include zero discharge months. No discharge during July, August, September and November 1980.

TABLE VIII-65. HISTORICAL DATA SUMMARY; BAUXITE MINE 5101; FINAL DISCHARGE FROM MINE AREA RUNOFF AND MINE PIT PUMPAGE -- TREATED EFFLUENT; DISCHARGE 001; JUNE 1978 - DECEMBER 1980

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS	MEAN†	STANDARD DEVIATION	RANGE
Flow (m^3/day)††	June 1978 - Dec. 1980	Continuous	–	6,540	2,900	0 - 12,536
TSS (mg/l)††	June 1978 - Dec. 1980	1/week	31	5.2	2.8	0 - 30.0
pH (Std. Units)	June 1978 - Dec. 1980	1/week	31	–	–	5.2 - 8.7
Aluminum (mg/l)††	Oct. 1978 - Dec. 1980	1-3/month	27	<0.36	0.32	<0.1 - 1.50
Iron (mg/l)††	Oct 1978 - Dec. 1980	1-3/month	27	<0.23	0.36	<0.01 - 5.2

*All metals expressed as total metals unless otherwise specified.

† Mean of monthly averages.

††Numbers rounded to significant figures.

TABLE VIII-66. HISTORICAL DATA SUMMARY; BAUXITE MINE 5101; FINAL DISCHARGE FROM MINE AREA RUNOFF AND MINE PIT PUMPAGE – TREATED EFFLUENT; DISCHARGE 007; JANUARY 1978 - DECEMBER 1980

PARAMETER**	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS	MEAN††	STANDARD DEVIATION	RANGE
Flow (m^3/day)*	Jan. 1578 - Dec. 1980	Continuous	–	370**	534**	0 - 4,360
TSS (mg/l)†	Jan. 1978 - Dec. 1980	1-4/month	16	6.8	5.0	0 - 32
pH (Std. Units)	Jan. 1978 - Dec. 1980	1/week	16	–	–	6.0 - 8.9
Aluminum (mg/l)†	Jan. 1978 - Dec. 1980	1-4/month	9	<0.15	0.06	<0.1 - 0.29
Iron (mg/l)†	Jan. 1978 - Dec. 1980	1-4/month	9	<0.76	0.60	<0.01 - 2.6

* Mean flow and standard deviation include zero discharge months.

**There were 20 months during the monitoring period in which no discharge occurred.

† Mean, standard deviation and range for discharging months only. Numbers rounded to significant figures.

††Mean of monthly averages.

TABLE VIII-67. HISTORICAL DATA SUMMARY; BAUXITE MINE 5101; FINAL DISCHARGE FROM MINE AREA RUNOFF AND MINE PIT PUMPAGE – TREATED EFFLUENT; DISCHARGE 009; JANUARY 1980 - SEPTEMBER 1980

PARAMETER**	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS	MEAN†	STANDARD DEVIATION	RANGE
Flow (m^3/day)*	Jan. 1980 - Sept. 1980	Continuous	–	1,060	1,050	0 - 8,176
TSS (mg/l)††	Jan. 1980 - Sept. 1980	3-4/month	7	5.9	2.6	0 - 26
pH (Std. Units)	Jan. 1980 - Sept. 1980	1/week	7	–	–	6.1 - 8.2
Aluminum (mg/l)††	Jan. 1980 - Sept. 1980	1-2/month	7	<0.27	0.14	<0.1 - 0.64
Iron (mg/l)††	Jan. 1980 - Sept. 1980	1-2/month	7	0.27	0.14	<0.05 - 0.53

* Mean flow and standard deviation include zero discharge months. No discharge occurred during January or September 1980.

**All metals expressed as total metals unless otherwise specified.

† Mean of monthly averages.

††Mean, standard deviation and range for discharging months only. Numbers rounded to significant figures.

TABLE VIII-68. HISTORICAL DATA SUMMARY FOR TUNGSTEN MINE 6104 (TREATED EFFLUENT)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Jan. 75 - Dec. 76	1/mo. (mo. ave.)	35	7.98 (ave. of mo. means)	0.27	7.3 – 8.6
TSS	Jan. 75 - Dec. 76	1-4/mo. (mo. ave.)	24	10.8 (ave. of mo. means)	4.4	4 – 20
Cu	Jan. 75 - Dec. 76	1/mo.	24	< 0.02	na	none
Mo	Jan. 75 - Dec. 76	1/mo.	24	0.074	0.17	< 0.02 – 0.74
Zn	Jan. 75 - Dec. 76	1/mo.	24	< 0.01	na	none

*All metals expressed as total metals unless otherwise specified.
na = not applicable

TABLE VIII-69. HISTORICAL DATA SUMMARY FOR URANIUM MINE 9408 (TREATED MINE WATER)

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATION	NUMBER OF OBSERVATIONS	MEAN (mg/l)	STANDARD DEVIATION (mg/l)	RANGE (mg/l)
pH	Apr. 75 - Jan. 77	1/mo.	21	8.3	0.3	7.8 – 8.7
TSS	Apr. 75 - Jan. 77	1/mo.	18	6	4	1 – 10
Ra226 (dissolved)	Apr. 75 - Jan. 77	1/mo.	19	0.55	0.45	0 – 1.2
U238	Apr. 75 - Jan. 77	1/mo.	21	2.0	1.3	0.2 – 4.2
Ba	Apr. 75 - Jan. 77	1/mo.	21	0.52	0.38	0.06 – 1.6
Mo	Apr. 75 - Jan. 77	4/yr.	7	0.65	0.53	0.05 – 1.3
Pb	Apr. 75 - Jan. 77	1/mo.	19	0.06	0.11	0.01 – 0.50
Zn	Apr. 75 - Jan. 77	4/yr.	7	0.02	0.02	0.008 – 0.08

*All metals expressed as total metals unless otherwise specified.

TABLE VIII-70. HISTORICAL DATA SUMMARY FOR NICKEL ORE MINE/MILL 6106 – TREATED EFFLUENT; DISCHARGE 001; JANUARY 1976 - DECEMBER 1980

PARAMETER†	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS	MEAN††	STANDARD DEVIATION	RANGE
Flow (m^3/day)*	Jan. 1976 - Dec. 1980	Daily	59	3,520**	5,560**	0 - 67,700
pH (Std. Units)***	Jan. 1976 - Dec. 1980	1/week	31	8.5	0.30	8.1 - 9.59
TSS (mg/l) ***	Jan. 1976 - Dec. 1980	1/week	32	18.6	15.4	0.4 - 138
Chromium (mg/l)***	Jan. 1976 - Dec. 1980	1/month	32	0.034	0.017	Not available
Manganese (mg/l) ***	Jan. 1976 - Dec. 1980	1/month	32	0.023	0.016	Not available
Phosphorous (P) (mg/l) ***	Jan. 1976 - Dec. 1980	1/month	31	0.05	0.017	Not available
Settleable Solids (mg/l) ***	Jan. 1976 - Dec. 1980	1/week	30	<0.05	0	<0.05 - 0.06

* Mean flow and standard deviation include zero discharge months.

**There are 28 months during the monitoring period in which no discharge occurred.

† All metals expressed as total metals unless otherwise specified.

††Mean of monthly averages.

***Mean, standard deviation and range for discharging months only. Number rounded to significant figures.

TABLE VIII-71. HISTORICAL DATA SUMMARY; VANADIUM MINE 6107; MINE AREA RUNOFF, WASTE PILE RUNOFF; DISCHARGE 005; JULY 1978 - DECEMBER 1980

PARAMETER*	MONITORING PERIOD	FREQUENCY OF OBSERVATIONS	NUMBER OF OBSERVATIONS	MEAN††	STANDARD DEVIATION	RANGE
Flow (m^3/day)**	July 1978 - Dec. 1980	Continuous	31	6,140	5,760	0 - 51,098
TSS(mg/l)***	July 1978 - Dec. 1980	1/week	27†	4	3	<1 - 12
Iron (mg/l)***	July 1978 - Dec. 1980	1/week	27†	0.65	0.23	0.15 - 2.9
pH (Std. Units)	July 1980 - Dec. 1980	5/week	28	–	–	5.4 - 9.3

* All metals expressed as total metals unless otherwise specified.

** Mean flow and standard deviation include zero discharge months. No discharge during September 1979, July 1980 and September 1980.

***Mean, standard deviation and range for discharging months only. Numbers rounded to significant figures.

† No values recorded for April 1980.

†† Mean of monthly averages.

TABLE VIII-72. HISTORICAL DATA SUMMARY FOR TITANIUM MINE/MILL 9906; TREATED MINE/MILL WATER; OCTOBER 1975 - DECEMBER 1979*

PARAMETER	MONITORING PERIOD	NUMBER OF OBSERVATIONS	MEAN**	STANDARD DEVIATION	RANGE
Flow (m^3/day)	Oct. 1975 - Dec. 1979	54	25,927	9,176	1,060 - 68,130
pH (Std. Units)	Oct. 1975 - Dec. 1979	54	7.0	–	4.0 - 10.0
TSS (mg/l)	Oct. 1977 - Dec. 1979	27	8.1	1.1	2.4 - 22
Settleable Solids (mg/l)	Oct. 1979 - Dec. 1979	3	<0.05	–	–

* Summarized from Mine/Mill 9906 Discharge Monitoring Reports from October 1975 - December 1979

**Mean of monthly averages

TABLE VIII-73. CHARACTERIZATION OF RAW WASTEWATER (TAILING-POND EFFLUENT AS INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT COPPER MILL 2122 DURING PERIOD OF 6-14 SEPTEMBER 1978

POLLUTANT PARAMETER	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
		MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	26	8.0	7.8-8.3	--	--
TSS	27	2554	19-44,600	--	--
Sb	23	<0.5	<0.5	--	--
As	↓	0.10	0.009-1.8	--	--
Be		<0.002	<0.002	--	--
Cd		0.014	0.010-0.040	0.012	0.010-0.015
Cr		0.19	0.025-3.7	--	--
Cu		2.0	0.010-43	0.027	0.020-0.035
Pb		0.16	0.050-1.9	0.068	0.050-0.090
Hg		0.0007	<0.0002-0.0020	--	--
Ni		0.19	<0.02-3.6	--	--
Se		0.022	0.007-0.20	--	--
Ag		0.014	<0.01-0.11	--	--
T1		<0.02	<0.02	--	--
Zn		0.10	0.015-1.9	0.007	<0.005-0.010
TOC	3	8	8-9	--	--
phenol	2	0.032	0.028-0.036	--	--

Based on observations made in period 6 through 14 September 1978.

TABLE VIII-74. CHARACTERIZATION OF RAW WASTEWATER (INFLUENT TO PILOT-SCALE TREATMENT TRAILER) AT BASE AND PRECIOUS METALS MILL 2122 DURING PERIOD OF 8-19 JANUARY, 1979.

POLLUTANT PARAMETER	PERIOD OF OBSERVATIONS (DATES)	NUMBER OF OBSERVATIONS	"TOTAL" CONCENTRATION (mg/l)		"DISSOLVED" CONCENTRATION (mg/l)	
			MEAN $\bar{x}$	RANGE	MEAN $\bar{x}$	RANGE
pH	Jan. 8-19, '79	16	8.8	8.5-8.9	--	--
TSS		10	213	25-1200	--	--
Sb		--	--	--	--	--
As		3	0.03	0.006-0.08	--	--
Be		--	--	--	--	--
Cd		--	--	--	--	--
Cr		3	0.07	0.04-0.13	<0.02	<0.02
Cu		3	0.44	0.04-1.24	0.08	0.02-0.21
Pb		3	0.11	0.06-0.19	0.06	0.05-0.08
Hg		--	--	--	--	--
Ni		3	0.06	0.02-0.14	0.02	0.02
Se		3	0.025	0.02-0.03	--	--
Ag		3	0.04	<0.03-0.08	0.03	0.03
T1		--	--	--	--	--
Zn		3	0.03	0.01-0.08	0.01	<0.01-0.03
Fe		3	23	0.42-68	0.04	0.02-0.07
COD		5	39	32-52	--	--
TOC		13	17	6-24	--	--
CN		9	0.03	0.003-0.060	--	--
Total Phenolics		14	0.58	0.23-0.81	--	--

TABLE VIII-75. SUMMARY OF PILOT-SCALE TREATABILITY STUDIES PERFORMED AT MILL 2122 DURING PERIOD OF 6-14 SEPTEMBER 1978

UNIT TREATMENT PROCESS EMPLOYED	EFFLUENT CONCENTRATION* ATTAINED (mg/l)						
	pH†	TSS	Cr	Cu	Pb	Ni	Zn
Sedimentation (2.8-hr theor. retention time)	7.9	50	0.035	0.05	0.09	0.07	0.03
Sedimentation (10.4-hr theor. retention time)	7.7	18	0.035	0.045	0.08	0.04	0.05
Polymer Addition, Lime Addition Flocculation, Sedimentation (2.8-hr theor. retention time)	9.3	21	0.04	0.04	0.09	0.05	0.03
Polymer Addition, Lime Addition Flocculation, Sedimentation (2.8-hr theor. retention time), Filtration	9.1** (9.0 to 9.2)	<1** (<1)	0.03** (0.03)	0.033** (0.03 to 0.04)	0.07** (0.06 to 0.09)	0.047** (0.04 to 0.05)	0.027** (0.025 to 0.030)
Polymer Addition, Lime Addition Flocculation, Sedimentation (2.8-hr theor. retention time),	9.9	52	0.035	0.035	0.06	0.04	0.02
Polymer Addition, Lime Addition Flocculation, Sedimentation (2.8-hr theor. retention time) Filtration	9.9	1	0.035	0.02	0.06	0.05	0.02
Filtration	8.0** (7.8 to 8.2)	7.5** (<1 to 30)	0.03** (0.02 to 0.04)	0.032** (0.01 to 0.055)	0.075** (0.05 to 0.11)	0.05** (<0.02 to 0.11)	0.06** (0.03 to 0.18)

* All metals concentrations are based on "total" analyses

† Value in pH units

**Average concentrations attained

( ) Range of concentrations attained

TABLE VIII-76. PERFORMANCE OF A DUAL MEDIA FILTER WITH TIME-FILTRATION OF TAILING POND DECANT AT MILL 2122

Hydraulic Loading on Filter = 762 m^3/m^2/day (13 gpm)
Initial TSS concentration = 33 mg/l

Time Elapsed	Final TSS Concentration mg/l
t_o + 15 min	12
t_o + 2 hr 15 min	7
t_o + 4 hr 15 min	11
t_o + 7 hr 15 min	23
t_o + 10 hr 15 min	31

TABLE VIII-77. RESULTS OF ALKALINE CHLORINATION BUCKET TESTS FOR DESTRUCTION OF PHENOL AND CYANIDE IN MILL 2122 TAILING POND DECANT*

*Unspiked cyanide concentration = < 0.01-0.06 mg/l
*Initial phenol concentration = 0.232-0.808 mg/l
*1 mg/l cyanide added to wastwater as NaCN

NaOCl Dose 5 mg/l

pH	9		10		11	
Contact Time (min)	CN^-	Total Phenolics	CN^-	Total Phenolics	CN^-	Total Phenolics
0	0.189	-	0.159	0.536	-	-
15	0.20	-	0.080	0.648	0.462	0.776
30	0.020	-	0.55	0.592	0.484	0.704
60	0.095	-	0.051	-	0.505	-
120	0.040	-	0.050	-	0.386	-

NaOCl Dose 10 mg/l

pH	9		10		11	
Contact Time (min)	CN^-	Total Phenolics	CN^-	Total Phenolics	CN^-	Total Phenolics
0	0.190	-	2.95	0.664	-	-
15	0.095	-	3.29	0.488	0.294	0.768
30	0.080	-	4.180	0.536	0.284	0.488
60	0.079	-	4.170	-	0.396	-
120	0.088	-	2.850	-	0.305	-

NaOCl Dose 20 mg/l

pH	9		10		11	
Contact Time (min)	CN^-	Total Phenolics	CN^-	Total Phenolics	CN^-	Total Phenolics
0	NOT PERFORMED		0.750	0.808	1.09	0.608
15			0.012	0.680	0.039	0.452
30			0.003	0.396	0.012	0.696
60			0.001	-	0.011	-
120			0.001	-	0.011	-

NaOCl Dose 50 mg/l

pH	9		10		11	
Contact Time (min)	CN^-	Total Phenolics	CN^-	Total Phenolics	CN^-	Total Phenolics
0	0.88	0.336	1.2	0.720	NOT PERFORMED	
15	0.05	0.228	0.001	0.058		
30	0.02	0.088	0.008	0.084		
60	0.01	-	0.003	0.080		
120	0.004	0.216	< 0.001	-		

All cyanide and phenol concentrations are mg/l.

TABLE VIII-78. RESULTS OF PILOT-SCALE OZONATION FOR DESTRUCTION OF PHENOL AND CYANIDE IN MILL 2122 TAILING POND DECANT*

O_3 Dosage		Initial pH	Retention Time	Final Concentration (mg/l)	
mg/min	Ratio O_3/CN			CN	Total Phenolics
2.4	2:1	10	10	0.610	0.416
0.8	2:1	9	30	0.460	0.256
6	5:1	10	10	0.035	0.312
2	5:1	11	30	0.040	0.428
1	5:1	10	60	0.016	0.272
24	10:1	10	5	-	0.052
8	10:1	9	15	0.020	0.026
4	10:1	10	30	0.065	0.200
2	10:1	8.5	60	0.035	0.474

*Initial Phenol Conc. = 0.232-0.808 mg/l; Cyanide added as NaCN to provide an initial conc. of 1 mg CN/l of wastewater.

TABLE VIII-79. EFFLUENT QUALITY ATTAINED AT SEVERAL PLACER MINING OPERATIONS EMPLOYING SETTLING-POND TECHNOLOGY

MINE	DATE SAMPLED	INFLUENT					EFFLUENT				
		pH*	TSS (mg/l)	Settleable Solids (ml/l/hr)	As (mg/l)	Hg (mg/l)	pH*	TSS (mg/l)	Settleable Solids (ml/l/hr)	As (mg/l)	Hg (mg/l)
4142	July 14, 1977	–	–	1.5	–	–	–	2080	0.3	0.27	<0.0002
4141	July 20, 1977	–	–	17	–	–	–	120	<0.1	0.031	<0.0002
4114	July 14, 1977	–	24,000	1.8	–	–	–	<0.1	<0.1	–	–
4140	July 17, 1977	7.3	1,130	1.7	–	–	8.5	220	<0.1	0.057	<0.0002
4139	July 12, 1977	7.4	9,000	13	1.2	0.004	7.4	230	0.15	0.012	<0.0002
4136	Aug. 26, 1978	–	64,100	45	3.9	0.001	–	150	<0.1	<0.002	<0.0002
4135	Aug. 24, 1978	–	2,890	7.5	0.04	0.020	–	474	0.7 – 0.9	0.022	<0.0002
4126	Aug. 15, 1978	6.5	14,800	260	1.3	0.0002	6.8	76	<0.1	0.25	0.0002
4127	Aug. 15, 1978	6.7	39,900	550	5.0	0.0014	6.4	5,700	2.5	1.2	0.0005
4132	Aug. 18, 1978	6.6	1,540	1.6 – 2.0	0.05	<0.0002	6.5	1040	0.4 – 0.8	0.05	<0.0002
4133	Aug. 19, 1978	7.9	2,260	0.7 – 1.6	1.5	0.0002	7.7	170	<0.1	0.06	0.0002
4134	Aug. 21, 1978	–	–	1.6	–	–	–	1420	0.4	0.28	<0.0002

*Value in pH units

Figure VIII-1. POTABLE WATER TREATMENT FOR ASBESTOS REMOVAL AT LAKEWOOD PLANT, DULUTH, MINNESOTA

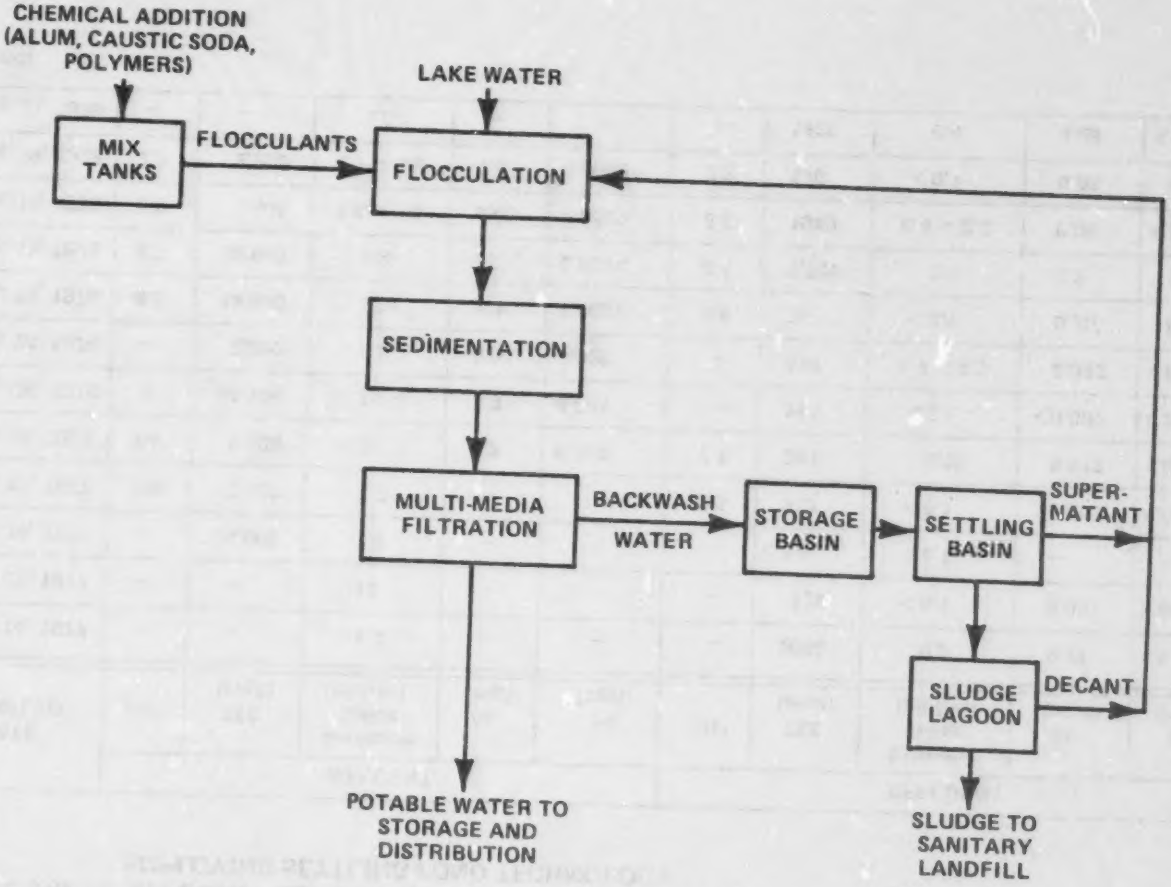

CHEMICAL ADDITION (ALUM, CAUSTIC SODA, POLYMERS)
LAKE WATER
MIX TANKS
FLOCCULANTS
FLOCCULATION
SEDIMENTATION
MULTI-MEDIA FILTRATION
BACKWASH WATER
STORAGE BASIN
SETTLING BASIN
SUPER-NATANT
SLUDGE LAGOON
DECANT
POTABLE WATER TO STORAGE AND DISTRIBUTION
SLUDGE TO SANITARY LANDFILL

Figure VIII-2. EXPERIMENTAL MINE-DRAINAGE TREATMENT SYSTEM FOR UNIVERSITY OF DENVER STUDY

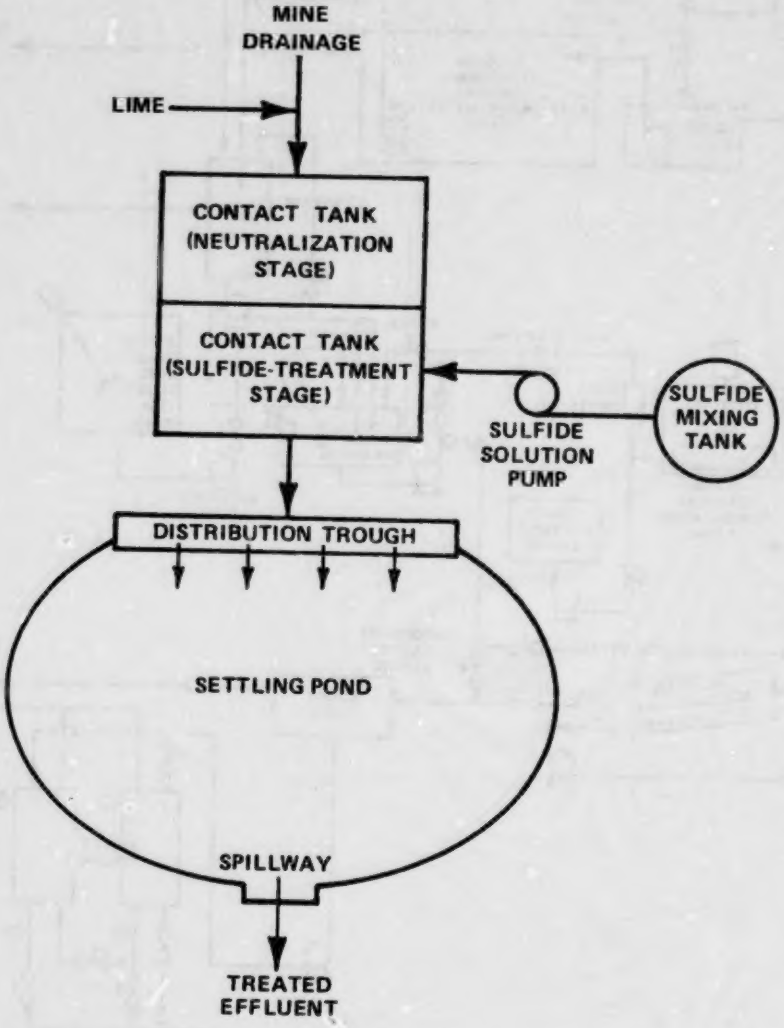
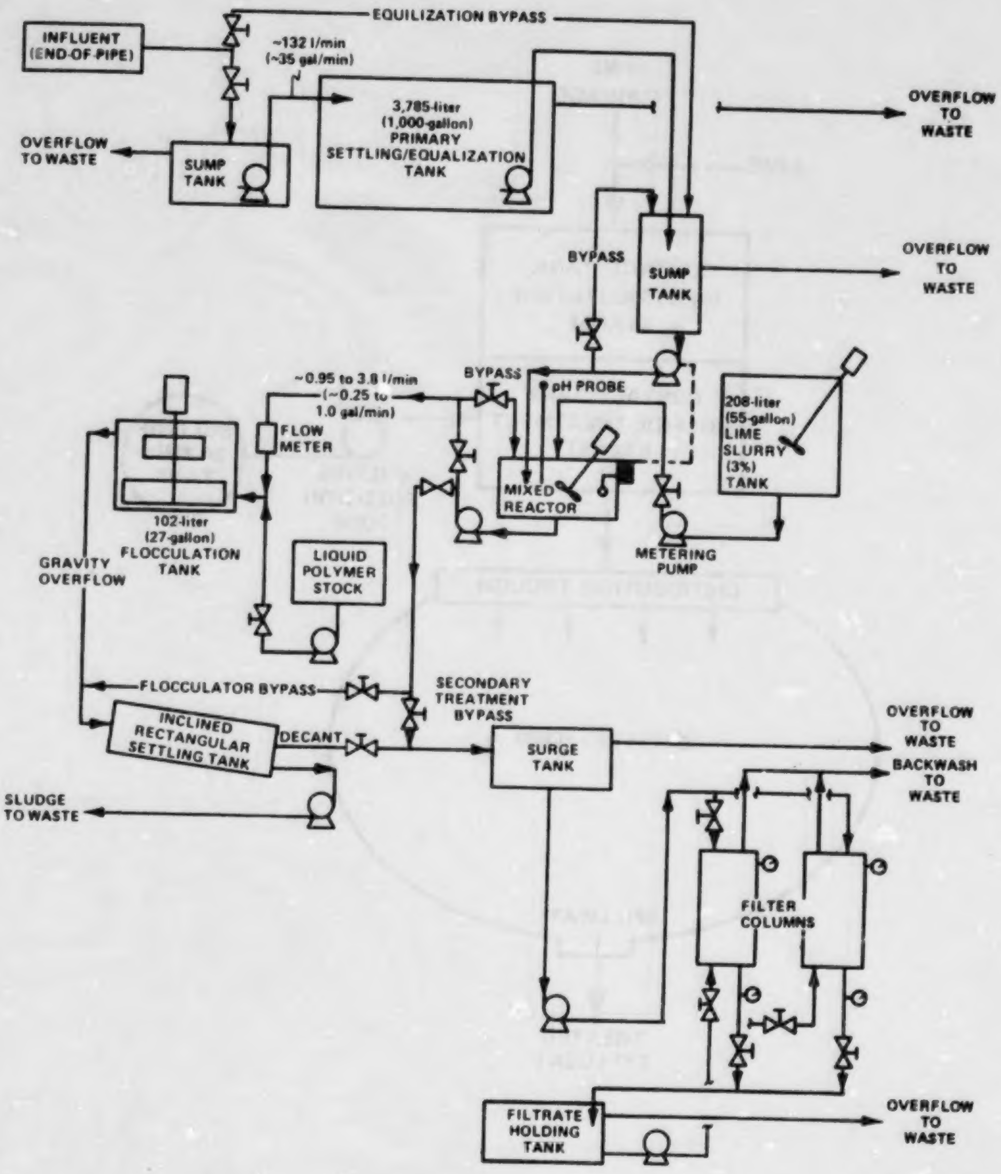
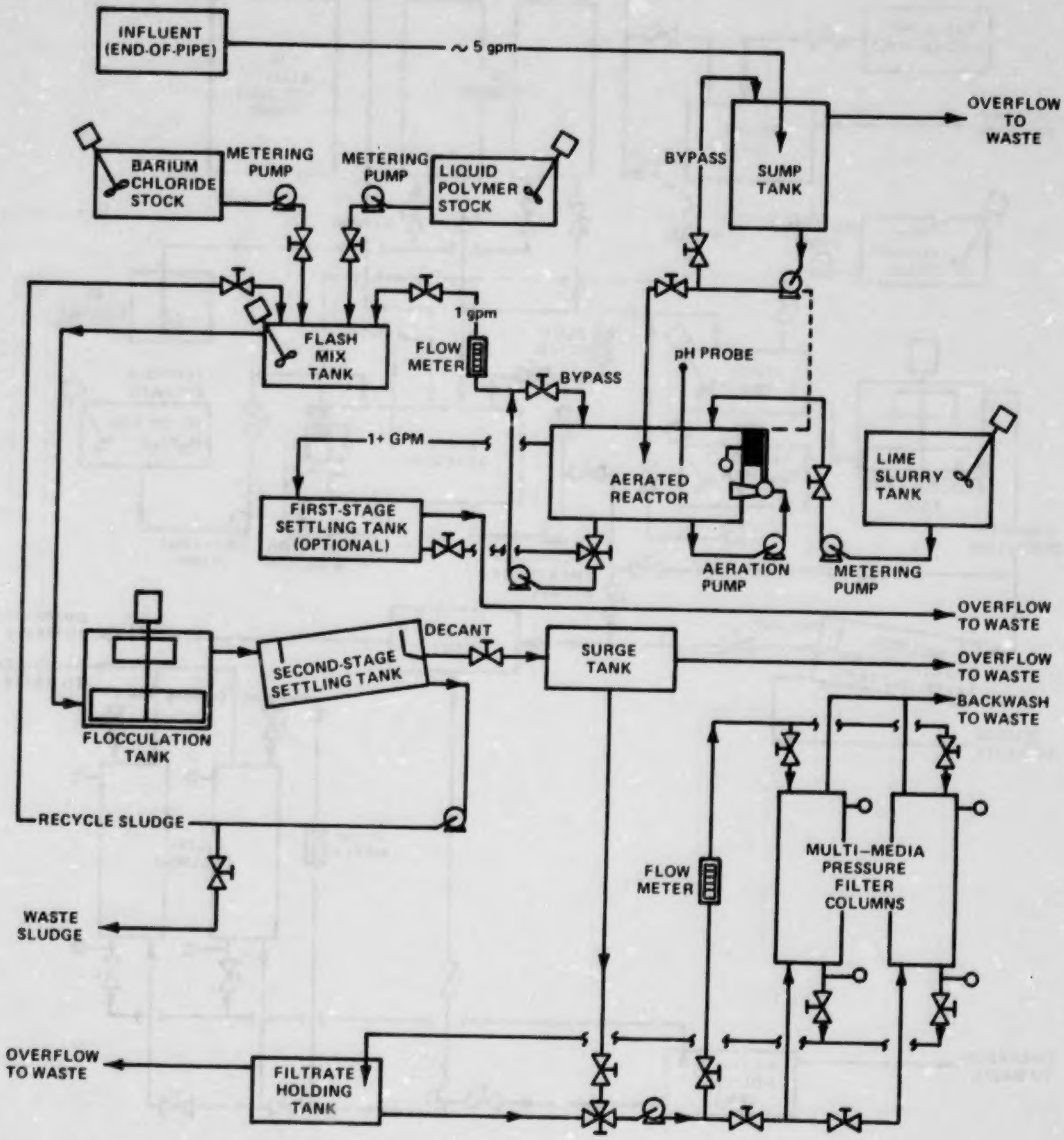

MINE DRAINAGE
LIME
CONTACT TANK (NEUTRALIZATION STAGE)
CONTACT TANK (SULFIDE-TREATMENT STAGE)
SULFIDE SOLUTION PUMP
SULFIDE MIXING TANK
DISTRIBUTION TROUGH
SETTLING POND
SPILLWAY
TREATED EFFLUENT

Figure VIII-3. CALSPAN MOBILE ENVIRONMENTAL TREATMENT PLANT CONFIGURATION EMPLOYED AT BASE AND PRECIOUS METAL MINE AND MILL OPERATIONS



INFLUENT (END-OF-PIPE)
EQUILIZATION BYPASS
~132 l/min (~35 gal/min)
OVERFLOW TO WASTE
SUMP TANK
3,785-liter (1,000-gallon) PRIMARY SETTLING/EQUALIZATION TANK
OVERFLOW TO WASTE
BYPASS
SUMP TANK
OVERFLOW TO WASTE
~0.95 to 3.8 l/min (~0.25 to 1.0 gal/min)
BYPASS
pH PROBE
FLOW METER
MIXED REACTOR
208-liter (55-gallon) LIME SLURRY (3%) TANK
METERING PUMP
102-liter (27-gallon) FLOCCULATION TANK
GRAVITY OVERFLOW
LIQUID POLYMER STOCK
FLOCCULATOR BYPASS
SECONDARY TREATMENT BYPASS
INCLINED RECTANGULAR SETTLING TANK
DECANT
SURGE TANK
OVERFLOW TO WASTE
BACKWASH TO WASTE
SLUDGE TO WASTE
FILTER COLUMNS
FILTRATE HOLDING TANK
OVERFLOW TO WASTE

Figure VIII-4. MOBILE PILOT TREATMENT SYSTEM CONFIGURATION EMPLOYED AT URANIUM MILL 9402



INFLUENT (END-OF-PIPE)
~ 5 gpm
OVERFLOW TO WASTE
BARIUM CHLORIDE STOCK
METERING PUMP
METERING PUMP
LIQUID POLYMER STOCK
BYPASS
SUMP TANK
FLASH MIX TANK
1 gpm
FLOW METER
BYPASS
pH PROBE
1+ GPM
AERATED REACTOR
LIME SLURRY TANK
FIRST-STAGE SETTLING TANK (OPTIONAL)
AERATION PUMP
METERING PUMP
OVERFLOW TO WASTE
DECANT
SECOND-STAGE SETTLING TANK
SURGE TANK
OVERFLOW TO WASTE
BACKWASH TO WASTE
FLOCCULATION TANK
RECYCLE SLUDGE
WASTE SLUDGE
FLOW METER
MULTI–MEDIA PRESSURE FILTER COLUMNS
OVERFLOW TO WASTE
FILTRATE HOLDING TANK

Figure VIII-5. PILOT TREATMENT SYSTEM CONFIGURATION EMPLOYED AT URANIUM MILL 9401

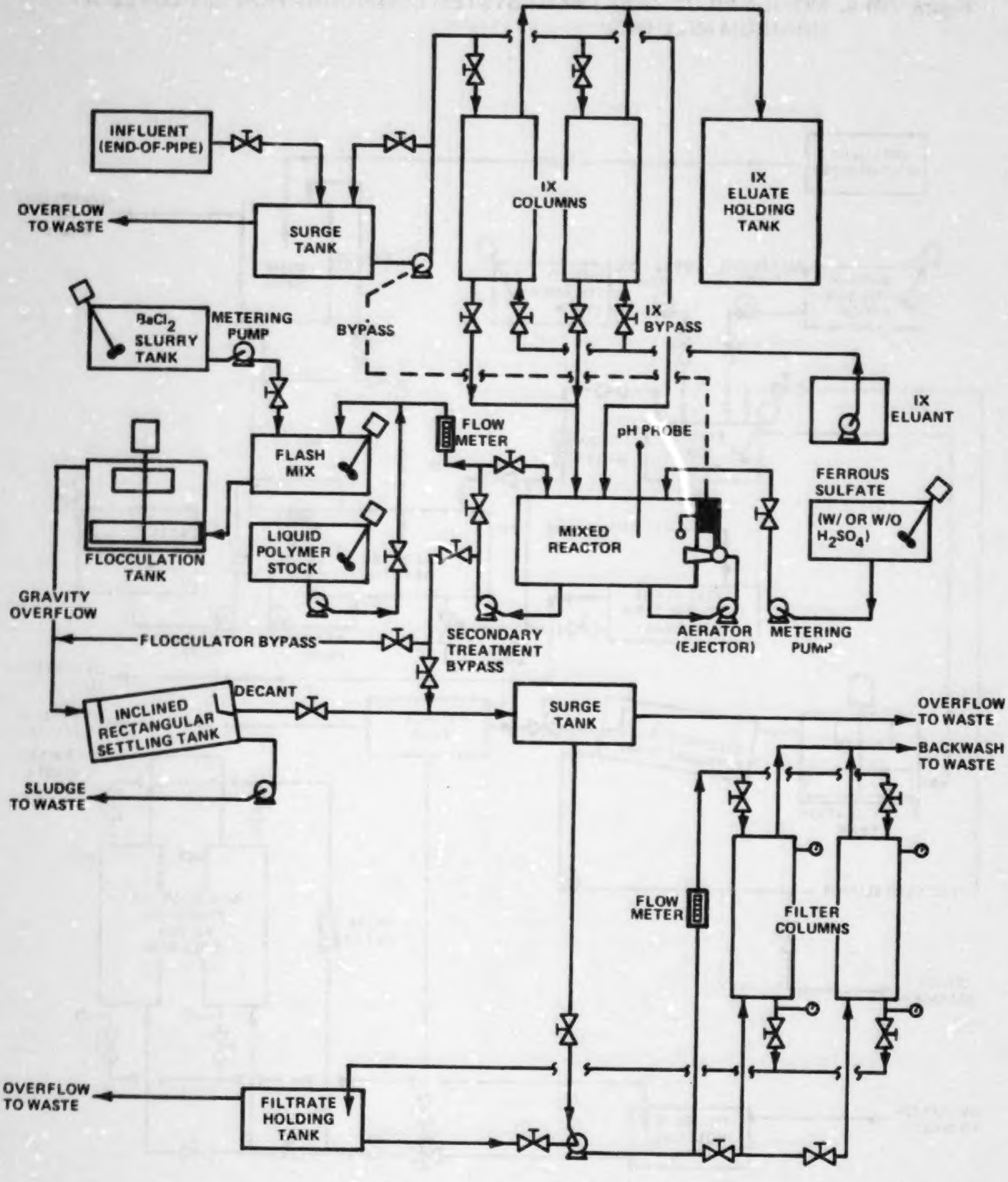

INFLUENT (END-OF-PIPE)
OVERFLOW TO WASTE
SURGE TANK
IX COLUMNS
IX ELUATE HOLDING TANK
BaCl$_2$ SLURRY TANK
METERING PUMP
BYPASS
IX BYPASS
IX ELUANT
FLASH MIX
FLOW METER
pH PROBE
FLOCCULATION TANK
LIQUID POLYMER STOCK
MIXED REACTOR
FERROUS SULFATE (W/ OR W/O H$_2$SO$_4$)
GRAVITY OVERFLOW
FLOCCULATOR BYPASS
SECONDARY TREATMENT BYPASS
AERATOR (EJECTOR)
METERING PUMP
DECANT
INCLINED RECTANGULAR SETTLING TANK
SURGE TANK
OVERFLOW TO WASTE
BACKWASH TO WASTE
SLUDGE TO WASTE
FLOW METER
FILTER COLUMNS
OVERFLOW TO WASTE
FILTRATE HOLDING TANK

Figure VIII-6. MODE OF OPERATION OF SEDIMENTATION TANK DURING PILOT-SCALE TREATABILITY STUDY AT MINE/MILL 9402

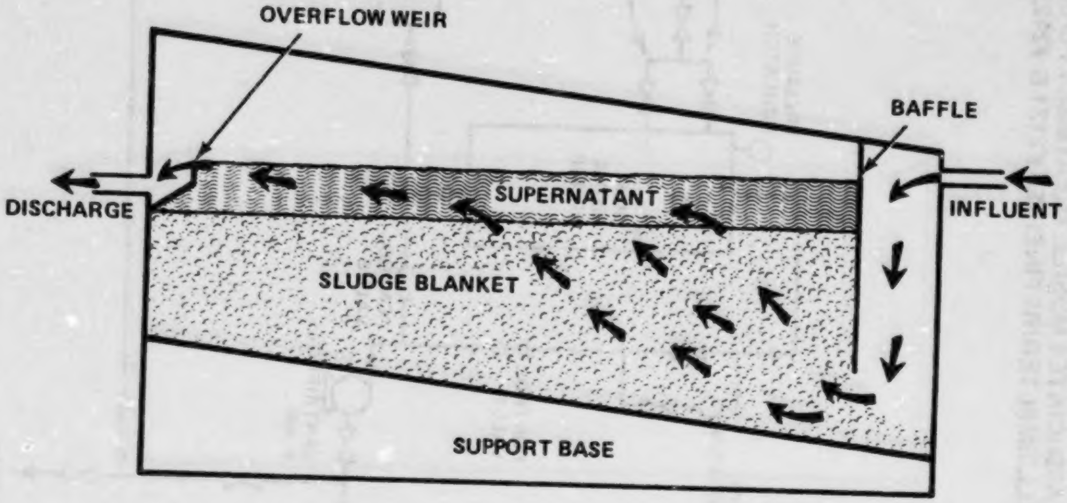
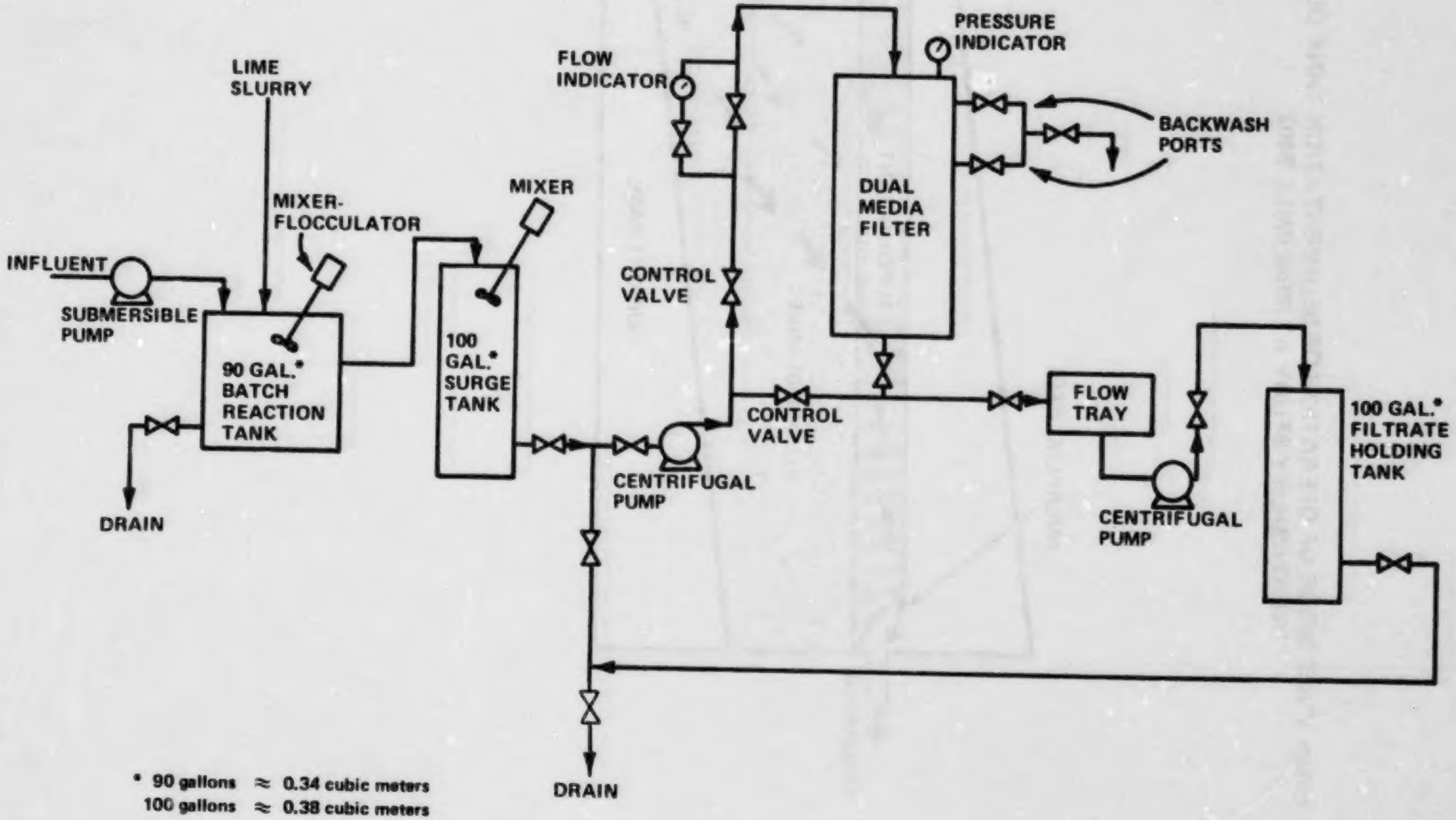
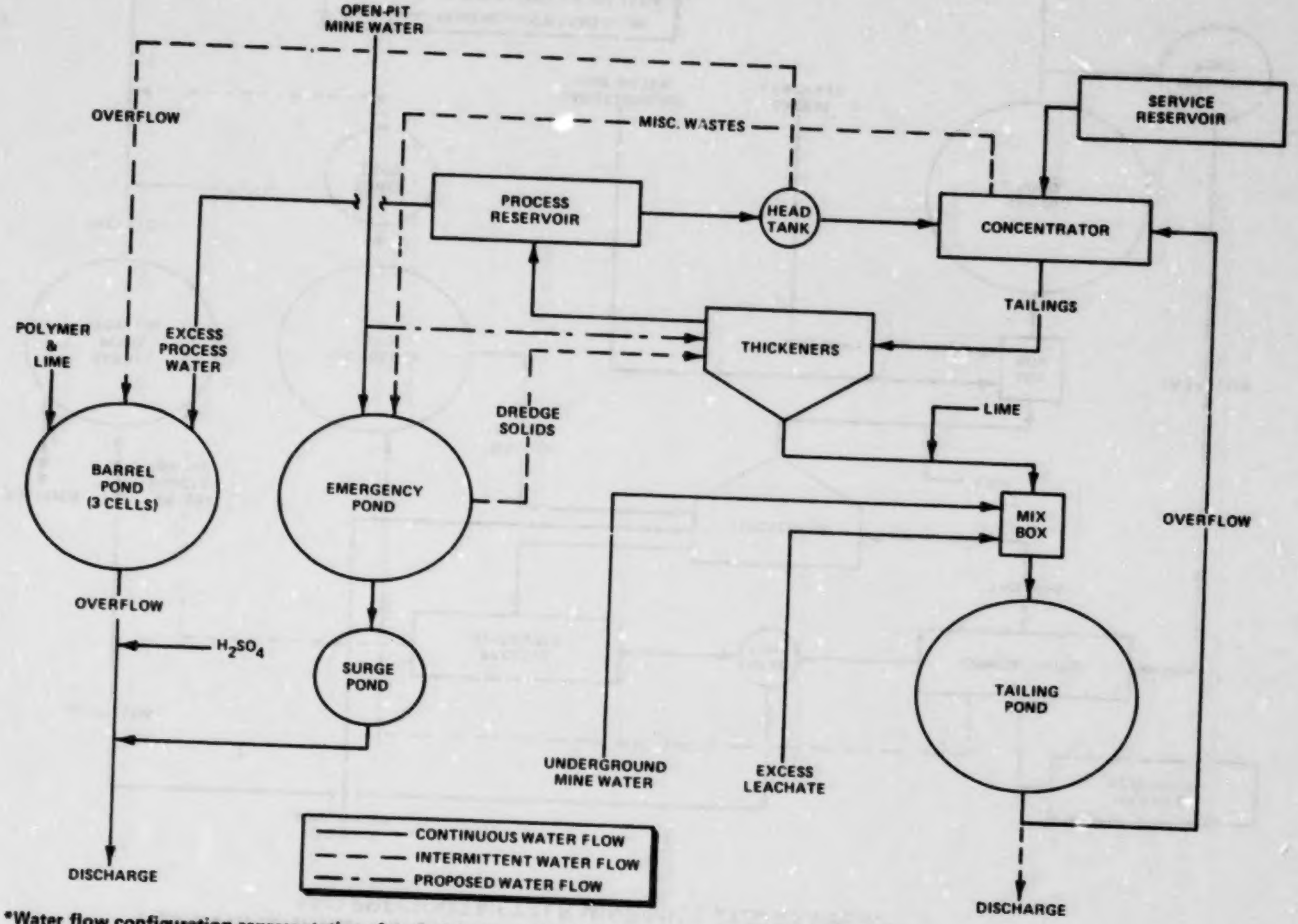

OVERFLOW WEIR
BAFFLE
DISCHARGE
SUPERNATANT
INFLUENT
SLUDGE BLANKET
SUPPORT BASE

Figure VIII-7. FRONTIER TECHNICAL ASSOCIATES MOBILE TREATMENT PLANT CONFIGURATION EMPLOYED AT MINE/MILL/SMELTER/REFINERIES #2121 & #2122


LIME SLURRY
MIXER-FLOCCULATOR
MIXER
INFLUENT
SUBMERSIBLE PUMP
90 GAL.* BATCH REACTION TANK
100 GAL.* SURGE TANK
DRAIN
FLOW INDICATOR
CONTROL VALVE
CONTROL VALVE
CENTRIFUGAL PUMP
PRESSURE INDICATOR
DUAL MEDIA FILTER
BACKWASH PORTS
FLOW TRAY
CENTRIFUGAL PUMP
100 GAL.* FILTRATE HOLDING TANK
DRAIN

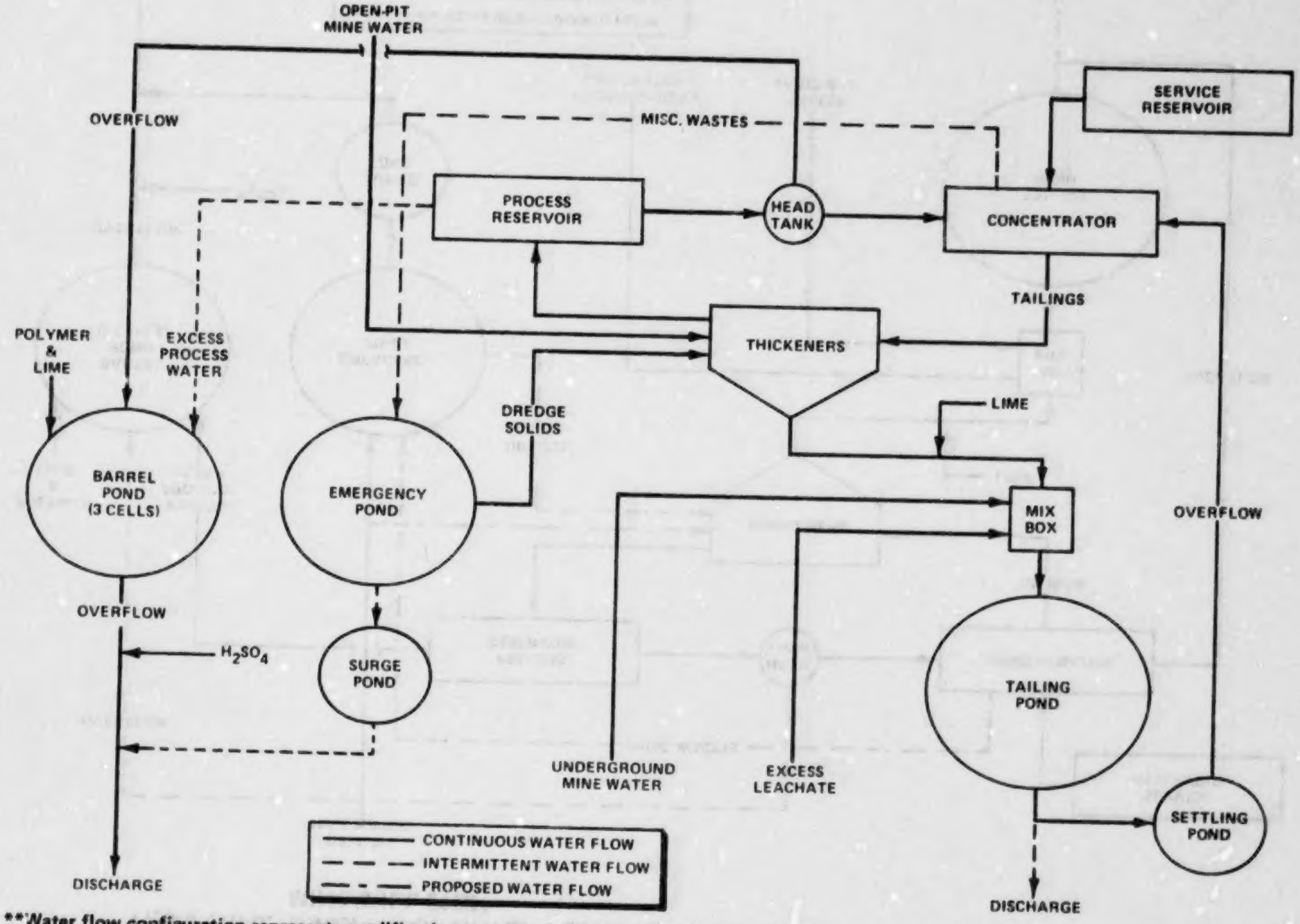
* 90 gallons ≈ 0.34 cubic meters
100 gallons ≈ 0.38 cubic meters

Figure VIII-8. DIAGRAM OF WATER FLOW AND WASTEWATER TREATMENT AT COPPER MINE/MILL 2120*


OPEN-PIT MINE WATER
OVERFLOW
MISC. WASTES
SERVICE RESERVOIR
PROCESS RESERVOIR
HEAD TANK
CONCENTRATOR
TAILINGS
THICKENERS
POLYMER & LIME
EXCESS PROCESS WATER
LIME
DREDGE SOLIDS
BARREL POND (3 CELLS)
EMERGENCY POND
MIX BOX
OVERFLOW
OVERFLOW
H_2SO_4
SURGE POND
TAILING POND
UNDERGROUND MINE WATER
EXCESS LEACHATE
CONTINUOUS WATER FLOW
INTERMITTENT WATER FLOW
PROPOSED WATER FLOW
DISCHARGE
DISCHARGE

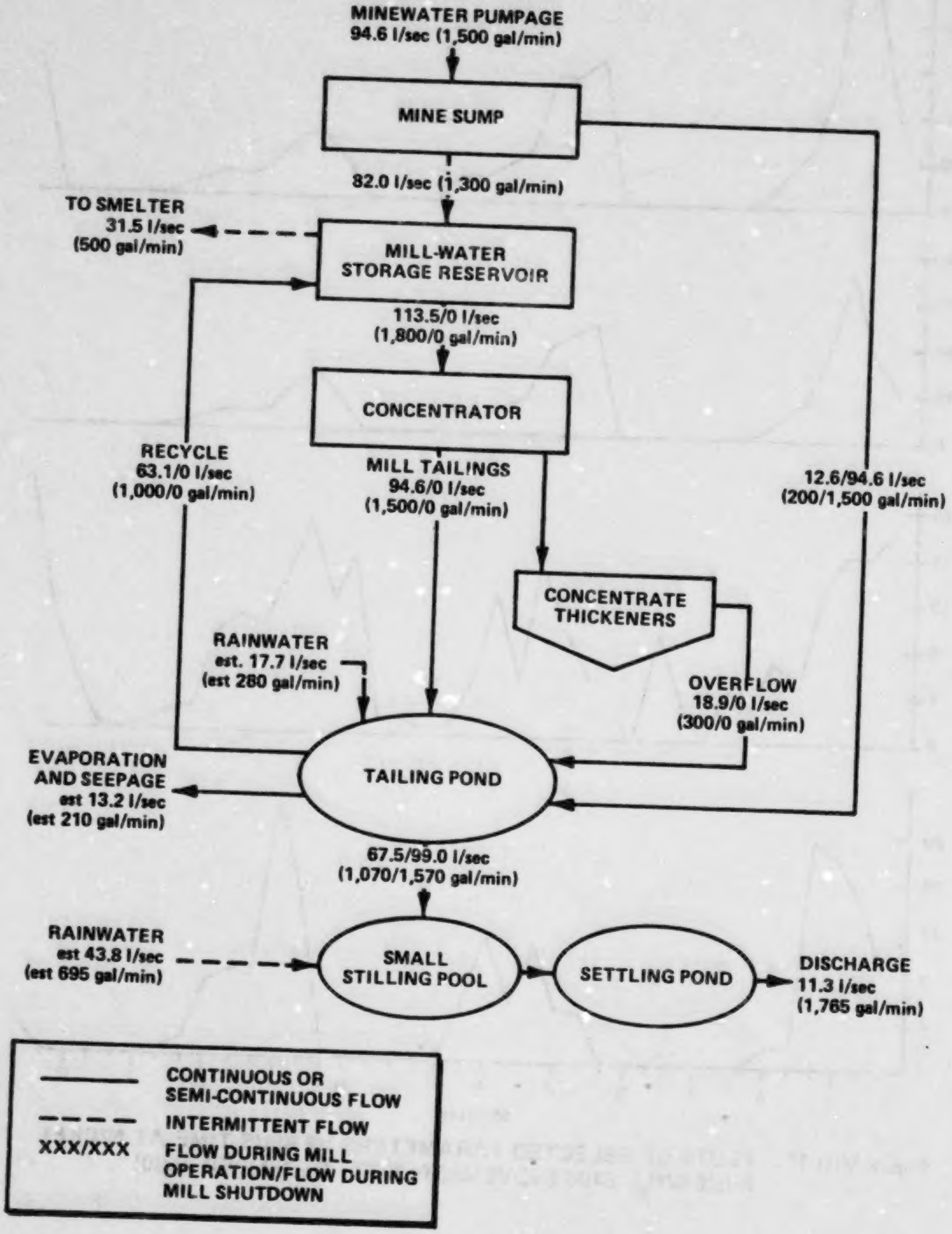
*Water flow configuration representative of period during which historical monitoring data was generated.

Figure VIII-9. DIAGRAM OF WATER FLOW AND WASTEWATER TREATMENT AT BASE AND PRECIOUS METALS MINE/MILL 2120 (COPPER)**


OPEN-PIT MINE WATER
OVERFLOW
MISC. WASTES
SERVICE RESERVOIR
PROCESS RESERVOIR
HEAD TANK
CONCENTRATOR
TAILINGS
POLYMER & LIME
EXCESS PROCESS WATER
THICKENERS
LIME
DREDGE SOLIDS
BARREL POND (3 CELLS)
EMERGENCY POND
MIX BOX
OVERFLOW
OVERFLOW
H_2SO_4
SURGE POND
TAILING POND
UNDERGROUND MINE WATER
EXCESS LEACHATE
SETTLING POND
CONTINUOUS WATER FLOW
INTERMITTENT WATER FLOW
PROPOSED WATER FLOW
DISCHARGE
DISCHARGE

**Water flow configuration represents modifications made to system as of September, 1979

Figure VIII-10. SCHEMATIC DIAGRAM OF WATER FLOWS AND TREATMENT FACILITIES AT LEAD/ZINC MINE/MILL 3103


MINEWATER PUMPAGE
94.6 l/sec (1,500 gal/min)
MINE SUMP
82.0 l/sec (1,300 gal/min)
TO SMELTER
31.5 l/sec
(500 gal/min)
MILL-WATER STORAGE RESERVOIR
113.5/0 l/sec
(1,800/0 gal/min)
CONCENTRATOR
RECYCLE
63.1/0 l/sec
(1,000/0 gal/min)
MILL TAILINGS
94.6/0 l/sec
(1,500/0 gal/min)
12.6/94.6 l/sec
(200/1,500 gal/min)
CONCENTRATE THICKENERS
RAINWATER
est. 17.7 l/sec
(est 280 gal/min)
OVERFLOW
18.9/0 l/sec
(300/0 gal/min)
EVAPORATION AND SEEPAGE
est 13.2 l/sec
(est 210 gal/min)
TAILING POND
67.5/99.0 l/sec
(1,070/1,570 gal/min)
RAINWATER
est 43.8 l/sec
(est 695 gal/min)
SMALL STILLING POOL
SETTLING POND
DISCHARGE
11.3 l/sec
(1,765 gal/min)
CONTINUOUS OR SEMI-CONTINUOUS FLOW
INTERMITTENT FLOW
XXX/XXX
FLOW DURING MILL OPERATION/FLOW DURING MILL SHUTDOWN

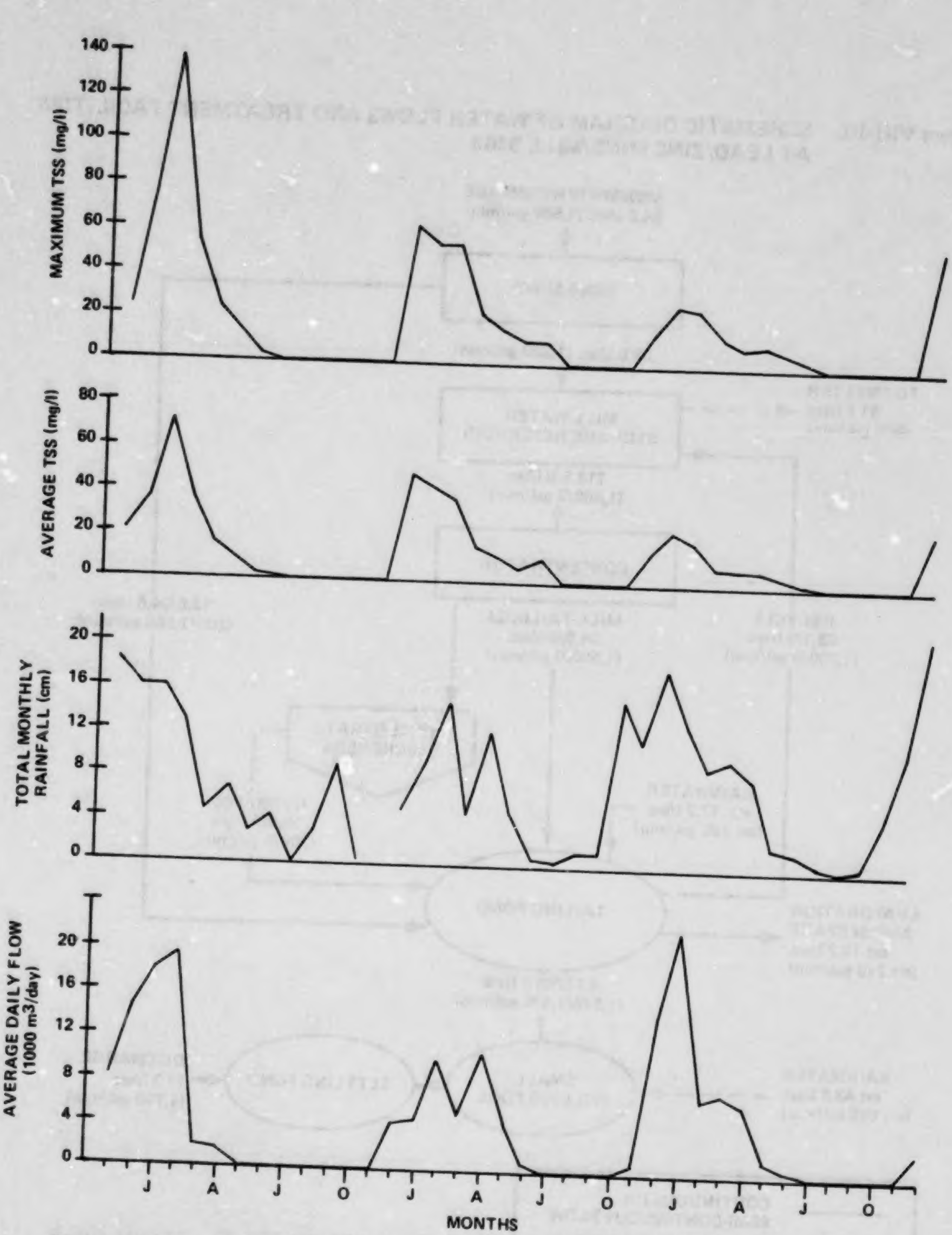
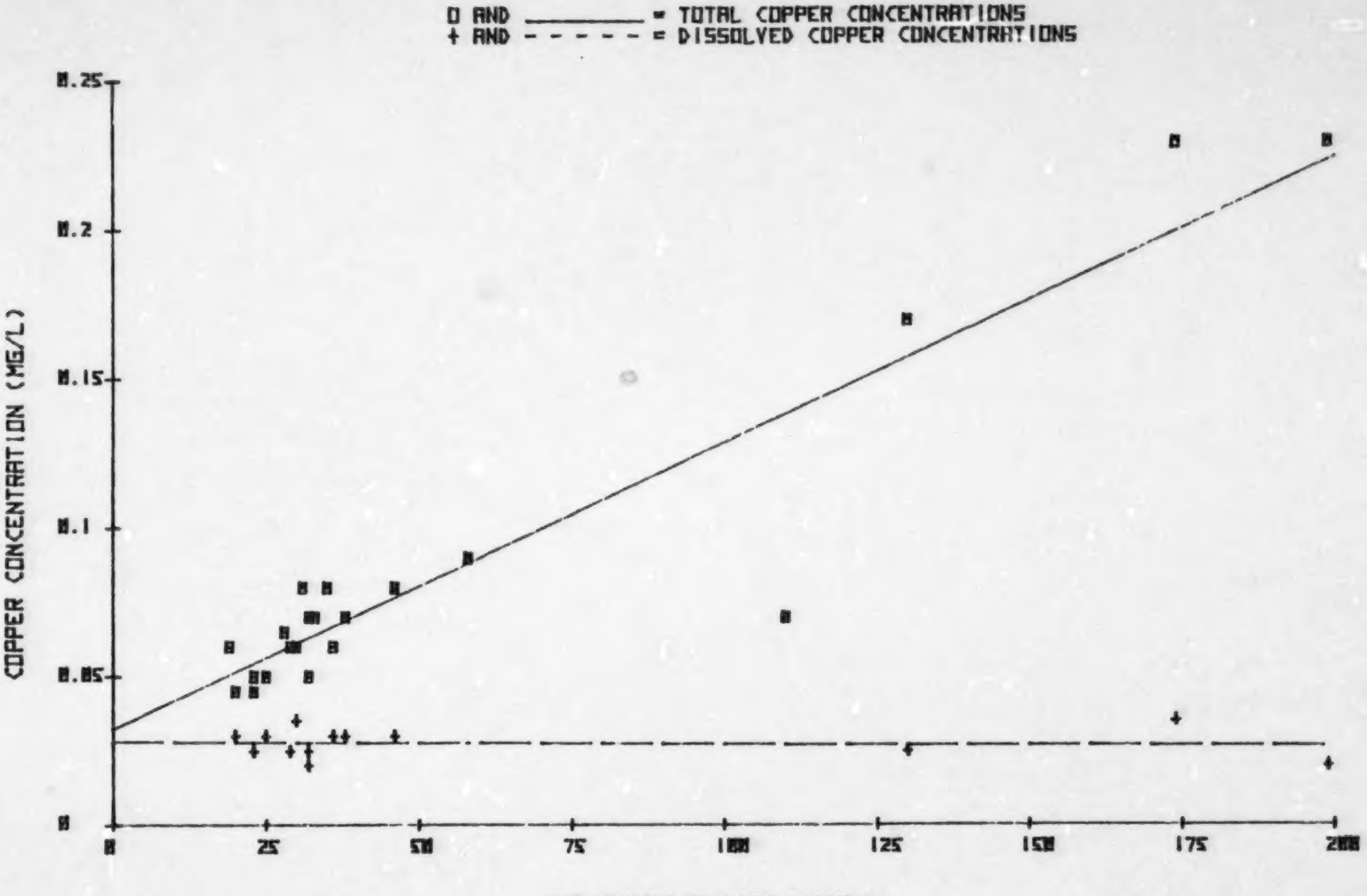


Figure VIII-11. **PLOTS OF SELECTED PARAMETERS VERSUS TIME AT NICKEL MINE/MILL 6106 (NOVEMBER 1977 - DECEMBER 1980)**

Figure VIII-12. PLOT OF TSS CONCENTRATIONS VS. COPPER CONCENTRATIONS IN TAILING-POND DECANT AT MINE/MILL/SMELTER/REFINERY 2122



SECTION IX

COST, ENERGY, AND NON-WATER QUALITY ISSUES

DEVELOPMENT OF COST DATA BASE

General

Generalized capital and annual costs for wastewater treatment processes at ore mining and dressing facilities have been established. Costing has been prepared on a unit process basis for each ore category. Assumptions regarding the costs, cost factors, and methods used to derive the capital and annual costs are documented in this section. All costs are expressed in 1979 dollars (Engineering News Record construction index=3140; 13 December 1979, Reference 1).

The estimates were based on assumptions pertaining to system loading and hydraulics, treatment process design criteria, and material, equipment, manpower, and energy costs. These assumptions are documented in detail in this section.

Fourth quarter 1979 vendor quotations were obtained for all major equipment and packaged systems. Construction costs were based on standard cost manual figures (see References 2 and 3) adjusted to December 1979.

The wastewater treatment processes studied are as follows:

Secondary Settling Ponds
Flocculation
Ozonation
Alkaline-Chlorination
Activated Carbon Adsorption
Hydrogen Peroxide Oxidation
Chlorine Dioxide Oxidation
Potassium Permanganate Oxidation
Ion Exchange
Granular Media Filtration
pH Adjustment
Recycle
Evaporation Ponds (total evaporation)

Table IX-1 indicates the processes studied for each ore subcategory.

CAPITAL COST

Capital Cost of Facilities

Settling Ponds. Construction costs for settling ponds were based upon assumptions (specifically documented later in this section)

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regarding the retention time and geometry of the ponds. Costs for excavation and back filling were assumed to be

Process Tankage. Mixing tanks, flocculation tanks, wet wells, ozone contactors and slurry tanks are sized for retention times appropriate to the particular process. These retention times are documented under the treatment process discussions later in this section. Construction cost estimates for tankage were then based on a factor of \$300/yd^3 of concrete (installed).

Reagent Storage Facilities. Cost estimates for tanks and bins used for reagent storage were based on vendor quotations. Sizing of the storage containers was based on dosage rates and backup supply assumptions which are documented in the treatment process discussions later in this section.

Buildings. Space requirements for housing treatment process equipment were based on vendor quotations. Building construction costs were developed from the methodology of References 2 and 3.

Piping. Unless otherwise stated, only local piping cost was included in the capital cost estimates, and installed costs were established from References 2 and 3. Long runs of interprocess piping have not been included due to their site-specific nature.

Lagoon and Tank Liners. Where required, lagoon or tank lining material was costed at two dollars per square foot (installed).

Structural Steel. Handrails and gratings, where required, were costed at one dollar per pound (installed) of fabricated steel equipment.

Capital Cost of Equipment

All equipment costs were obtained from vendor quotations. Instrumentation and electrical packages (installed) were assumed to be a percentage of the equipment costs. The percentages documented in the individual treatment process discussions later in this section, varied with the process in question.

Capital Cost of Installation

Unless otherwise stated, installation costs for equipment were included in the vendor quotations. Construction costs for facilities, including concrete, steel, ponds, tanks, piping and electrical, were estimated on an installed basis.

Capital Cost of Land

Land costs were estimated at \$4,000/acre unless otherwise stated.

Capital Cost of Contingency

Unless otherwise stated, a contingency cost of 20 percent was added to the total capital costs generated. This was intended to cover taxes, insurance, over-runs and other contingencies.

ANNUAL COST

Annual Cost of Amortization

Initial capital costs were amortized on the basis of a 10 percent annual interest rate with assumed life expectancy of 30 years for general civil and structural equipment and 10 years for mechanical and electrical equipment. Capital recovery factors were calculated using the formula:

$$CRF = \frac{(r)\,(1+r)^n}{(1+r)^n - 1}$$

where CRF = capital recovery factor
r = annual interest rate
and n = useful life in years.

Annual cost of amortization was computed as:

$$C_A = B\,(CRF)$$

where C_A = annual amortization cost

and B = initial capital cost.

Annual Cost of Operation and Maintenance

Maintenance. Annual maintenance costs were assumed to be three percent of the initial total capital cost unless otherwise noted.

Operation. Operating personnel wages were assumed to be $13.50/hr. including fringe benefits, insurance, etc. Estimated weekly operator manhours were established depending on the process and the hydraulic flow rate. These manpower estimates are documented in the individual treatment process discussions later in this section.

Reagents. The following prices were used to estimate annual costs of chemicals:

Polymer	$ 2.00/lb.
Sodium Hydroxide	$160.00/ton
Sodium Hypochlorite	$ 0.40/lb.

Hydrated Lime	$ 65.00/ton
Activated Carbon	$ 0.50/lb.
Hydrogen Peroxide (70% conc.)	$ 0.35/lb.
Sulfuric Acid (66 Be)	$ 0.04/lb
Ferrous Sulfate (400 lb. drum dry powder)	$ 0.52/lb.
Chlorine Dioxide (5% conc. in 55 gal. drum)	$ 9.10/gal.
Potassium Permanganate (dry)	$ 0.59/lb.

Reagent dosages are documented in the treatment process discussions in this section.

Annual Cost of Energy

The cost of electric power was assumed to be three cents per kilowatt-hour. Facilities were assumed to operate 24 hours per day, 365 days per year.

Monitoring Costs

Additional wastewater monitoring costs were estimated as $7,000 per year for ozonation and alkaline chlorination systems, and $10,000 per year for the remaining technologies except recycling. These figures were intended to account for those added monitoring costs associated only with the technologies described in this section.

TREATMENT PROCESS COSTS

Secondary Settling

Capital Costs. The cost of constructing settling ponds can vary widely, depending on local topographic and soil conditions. Figure IX-1 depicts the typical layout assumed for these estimates.

The costs and required sizes of settling ponds were developed as a function of hydraulic load. The basins were sized for a 24-hour retention time with an anticipated 10 percent safety factor (for sediment storage). It was assumed that lagoons and settling ponds are rectangular in shape, with the bottom length twice the bottom width. The dikes (berms) were constructed with a 2.5:1 slope. In all cases, the water depth was assumed to be 16 feet and a one-foot freeboard was provided. Water was presumed to flow by gravity.

For estimating purposes, it was assumed that 60 percent of the total basin volume required excavation and backfilling (estimated corrugated steel, and a total length of 200 feet was allowed (estimated cost: $17.30/ft.). However, it was recognized that longer runs of process interconnecting piping may be necessary in individual cases.

Complete capital cost estimates included costs for land, excavation and backfilling, piping, installation of piping, concrete pad for piping support, and pond liners (where necessary to prevent seepage). The capital cost curve in Figure IX-2 expresses the total capital cost as a function of hydraulic flow rate for secondary settling ponds. A contingency cost factor of 20 percent was included in these estimates. Figure IX-3 expresses the estimated settling ponds line cost.

Annual Costs. Annual maintenance costs for secondary settling ponds were assumed at $2,000, with additional monitoring costs of $10,000/year. Amortization was based on a 30-year life expectancy at 10 percent annual interest (CRF=0.10608). The annual costs displayed in Figure IX-2 as a function of hydraulic flow rate are the sum of the amortization, monitoring, and maintenance costs. Annual costs for pond liners are shown in Figure IX-3.

Flocculant Addition

Capital Costs. Capital costs were estimated for flocculation systems consisting of the equipment shown in Figure IX-4. A complete, installed mechanical package, which included the flocculant preparation and feed equipment, was based on vendor quotations. This package, designed for use with dry polymer, included storage tank, feeder, wetting equipment, aging tank with mixer, transfer pump, electrical and instrumentation package, and installation. Piping, tanks, and metering pumps were corrosion resistant. The remaining capital costs included site preparation, enclosure, and civil work (i.e., grading, concrete, super structure) as well as heating equipment (electrical heater, installed). In addition, the total capital cost included a 20 percent contingency cost factor.

The systems were sized based on hydraulic flow rate; consequently, total capital cost is expressed as a function of wastewater flow rate (Figure IX-5). A flocculant dosage of one part per million was used. A one- to five-minute mixing time, and a 30-day reagent storage capacity were assumed.

Local electrical and piping connections were included in the cost estimates. However, long runs of process interconnecting piping and electrical power lines, if necessary, will need to be estimated on a site-specific basis.

Annual Costs. Amortization of capital cost for flocculation systems assumed a 10 percent annual interest rate with life expectancies of 30 years for construction (CRF = 0.10608) and 10 years for mechanical and electrical equipment (CRF = 0.16275). Operator hours were estimated at 13.3 hours per week (1/3 time), and operator wages were calculated at $13.50 per hour including benefits. Additional cssts were estimated as follows: annual maintenance as three percent of capital cost; chemicals at a price of $2.00 per pound for dry polymer; energy at a rate of $0.03 per kilowatt-hour; and additional monitoring at $10,000 per year. An annual cost curve has been generated (Figure IX-5) expressing the total of the above expenses as a function of wastewater flow rate.

Ozonation

Capital Costs. The ozonation systems estimated in this section were defined by the flow diagram shown in Figure IX-6. The system equipment supply included air compressor with inlet filter/ silencer, after cooler, refrigerant cooling system, dessicant drying system, ozone generator, cooling tower, and concrete ozone contact chamber, located indoors near the contact chamber.

Equipment costs for the ozonation systems were based on vendor quotation. Building construction costs were based on vendor definition of special requirements with cost factors developed from References 2 and 3. Installation costs were based on the same references. A concrete cost factor of $300/yd³ (installed) served as a basis for the ozone contact chamber costs.

The ozonation system design estimates were based on an ozone dosage of five mg/l and a contact time of 15 minutes.

Total capital cost figures included equipment, installation, building construction, contactor tankage, and a 20 percent contingency factor. The capital cost graph in Figure IX-7 expresses the total capital cost as a function of flow in million gallons per day.

Operating Costs. Amortization of capital costs was based on a 10 percent annual interest rate, a 30-year life expectancy for construction (CRF = 0.10608), and a 10-year life expectancy for equipment (CRF = 0.16275). Maintenance costs were assumed to be three percent of the initial capital investment annually. Operator manhours were estimated at 20 hours per week for systems treating less than 10 million gallons per day, 30 hours per week for 10 to 100 million gallons per day systems, and 40 hours per week for systems treating greater than 100 million gallons per day of wastewater.

Operator wages were costed at $13.50/hour including benefits. Energy costs were based on a rate of $0.03 per kilowatt hour (3 cents). Electric power required for ozone generation was assumed to be 10 to 12 kwh per pound of ozone generated. The annual cost curve in Figure IX-7 depicts the sum of the above annual costs as a function of flow in million gallons per day. Monitoring costs of $7,000 per year should be added to the cost obtained from the curve.

Alkaline Chlorination

Capital Costs. The alkaline-chlorination system cost estimates were generated based on the use of sodium hydroxide and sodium hypochlorite as alkalinity and chlorine sources, respectively. System definition is represented by flow schematic in Figure IX-8.

Total capital cost estimates included storage facilities, mixing tank with liner, mixers, electrical and instrumentation package, reagent feed pumps, local piping and contingency costs (at 20 percent). Figure IX-9 includes a graph of total capital cost as a function of hydraulic flow rate.

Cost estimates for chemical storage tanks, chemical feed pumps, and mixing equipment were obtained from vendor quotations. The two chamber mixing tank was estimated at $300/yd³ installed; and electrical and instrumentation package costs were estimated at 20 percent of the total equipment cost.

In considering the capital costs, several system design assumptions were made, including:

1. Sodium hydroxide dosage of 30 mg/l and sodium hypochlorite dosage of 10 mg/l
2. Mixing tanks sized for a two-minute retention time
3. Reagent storage capacity sized for a 30-day supply of each chemical
4. Sodium hydroxide and sodium hypochlorite estimated to
5. Use of turbine-type mixers (carbon steel construction), reciprocating (plunger) chemical feed pumps, carbon steel sodium hydroxide handling and storage equipment, and fiberglass sodium hypochlorite handling and storage equipment

Annual Costs. Capital recovery was amortized over a 10-year period for equipment and a 30-year period for construction. A 10 percent annual interest rate was used for both equipment and construction. (Equipment CRF = 0.16275, Construction CRF = 0.10608). Annual maintenance costs were assumed to be three percent of the initial capital investment. Operator manhours were estimated at 10 hours/week and were costed at a rate of $13.50/hour including benefits. Energy costs were developed at a rate of $0.03/kilowatt hour; chemical costs were based on the dosages previously mentioned (30 mg/l NaOH; 10 mg/l NaOCl); chemical prices (delivered) were estimated at $160.00/ton (2,000 pounds) for caustic soda (NaOH) and $0.40/pound for sodium hypochlorite (NaOCl); and additional monitoring costs of $7,000/year were assumed. Figure IX-9 includes the annual cost curve.

Ion Exchange

Capital Costs. The flow schematic for the ion exchange system is exhibited in Figure IX-10. This is a combination cation-anion-mixed bed process with a pretreatment (filtration) step. The system costed consists of skid-mounted package units including raw waste filters in steel tanks, cation exchangers, a degasifier, anion exchanger, and mixed bed exchangers. Acid is provided for regeneration of cation exchangers and caustic soda for anion exchangers. These waste solutions are mixed and require disposal. (All units are housed in a structure.)

Total capital costs include equipment, installation, building construction, and 20 percent contingency. The capital cost curve in Figure IX-11 relates this total capital cost to hydraulic flow rate.

Supply and installation cost estimates for all equipment were obtained from vendor quotations. Units were sized for hydraulic loading according to vendor recommendations. Building construction costs (including concrete foundations) were estimated based on vendor space requirement quotes and the costing methodology of References 2 and 3.

Annual Costs. Amortization of initial capital investment was based on a 10 percent annual interest rate at a 10-year life expectancy for equipment (CRF = 0.16275) and a 30-year life expectancy for construction (CRF = 0.10608). Annual maintenance costs were estimated at three percent of initial capital cost. Reagents were costed at $0.11/pound for caustic soda and $0.03/pound for sulfuric acid. Electric power was costed at $0.03/KWH. Operator hours were estimated at 20 hours per week for plants treating less than 2.5 MGD, at 30 hours per week for plants treating 2.5 to 10.0 MGD, and at 40 hours per week for plants treating 10.0 to 35.0 OMGD. Operator wages and benefits were estimated to total $13.50/hour. Additional monitoring costs of $10,000/year were assumed. Figure IX-11 displays total annual costs as a function of daily flow rate.

Granular Media Filtration

Capital Costs. Figure IX-12 depicts the basic granular media filtration system proposed for cost estimates. Industrial, gravity flow deep bed, granular media filters were selected. The filters would be contained in prefabricated, portable steel filter units. Treated effluent would discharge through a concrete backwash wastewater basin where the filtered solids would settle. The supernatant would then be pumped back to the filters for treatment.

All piping is carbon steel, valves are the butterfly type, and the pumping equipment consists of vertical turbine pumps of carbon steel construction. Pump impellers are bronze with stainless steel shafts. Filter media consists of plastic filter bottom, gravel, sand, and anthracite.

Vendor quotations obtained for filters, pumps, and air blowers (for backwash) included site preparation, installation, local piping, and instrumentation and electrical package. Systems were sized for a hydraulic loading of 10 gpm/ft^2.

Total capital costs included a 20 percent contingency factor. Figure IX-13 displays capital cost as a function of daily wastewater flow.

Annual Costs. Initial capital investment was amortized at a 10 percent annual interest rate over a period of 10 years for equipment (CRF = 0.16275) and 30 years for construction (CRF = 0.10608).

Costs estimated under annual costs include: (1) maintenance estimated at three percent of annual capital cost; (2) operator manhours established at 20 hours per week for systems treating one to five million gallons per day (MGD) and 30 hours per week for systems treating 10 to 100 MGD of wastewater; (3) electricity computed at a rate of $0.03 KWH; and (4) additional monitoring at $10,000 per year. Figure IX-13 includes the annual cost curve for these systems.

pH Adjustment

Capital Costs. System costs for pH adjustment by hydrated lime addition were developed. A schematic representation of the system is displayed in Figure IX-14. Major system components include lime storage and feed equipment, slurry tanks, feed pump, mixing tankage, and mixing equipment. The dry lime is stored in a steel silo which is equipped with a screw type feeder. The feed ratios of lime and water are preset and are started based on level in the steel lime slurry tanks. A vertical type turbine pump will pump the slurry into the wastewater mixing tanks. The tanks are reinforced concrete structures containing 3 turbine mixers of carbon steel construction. Mixers are for tank top mounting.

Costs of lime storage and feed equipment as well as mixer costs were obtained from vendor quotations. Mixing tankage and lime slurry tankage costs were based upon installed costs of lined concrete tanks. Electrical and instrumentation package installed costs were assumed to be 20 percent of the equipment costs.

Cost estimates were completed based upon a 50 mg/l dosage of 93 percent hydrated lime. A 30-day supply of lime was assumed for the design of storage facilities. Lime slurry tankage was sized for a 24-hour detention time, while mixing tanks were sized for a two minute detention time for flows of up to 10 mgd, and a one minute detention time for flows greater than 10 mgd.

Total capital cost estimates included storage bins, feeder equipment, concrete and lining material for slurry and mixing tanks, mixers, slurry pumps, electrical and instrumentation, installation, and contingency (at 20 percent). Figure IX-15 represents total capital cost as a function of wastewater flow.

Annual Costs. Annual costs estimated for the pH adjustment process included the following: (1) amortization calculated at a 10 percent annual interest rate for a 10-year life expectancy for equipment (CRF = 0.16275) and a 30-year life expectancy for construction (CRF = 0.10608); (2) annual maintenance costs estimated at three percent of the initial capital investment; (3) operator manhours established at 10 hours per week and costed at \$13.50/ hour including benefits, insurance, etc; lime costs based on a price of \$65/ton (2,000 lbs.); (4) cost of energy estimated at \$0.03/KWH, (3 cents); and (5) additional monitoring costs of \$10,000/year. Figure IX-16 displays annual cost as a function of daily wastewater flow.

Recycle

Capital Costs. Cost estimates were prepared for installation of systems to provide for 25, 50, 75, and 100 percent recycle of wastewater. Figure IX-17 represents the equipment and tankage requirements on which the estimates were based. Recycle is accomplished by collecting the effluent wastewater in a concrete tank. Pumps are provided to return all or a portion of the flow back to the mine or mill operations for reuse. Any quantity greater than the recycle rate would overflow into the receiving stream.

Recycle pumps are vertical turbine type complete with weather proof motor for outdoor installations. Collection sewer and pump discharge piping were not included in the costing.

Pumping equipment costs were based on vendor quotations. Wet well costs were based on \$300/yd^3 installed concrete cost. Local piping, valves, and fittings were costed based on vendor definition and costing methodology taken from Reference 2. Structural steel requirements for railings, gratings, etc. were costed at a rate of one dollar per pound (installed). Electrical and instrumentation package costs (installed) were estimated at 30 percent of the total equipment cost.

Pumping equipment selection was based on hydraulic flow requirements assuming 75 feet total dynamic head requirement. Wet well sizing was based on a 10-minute retention time.

Total capital cost estimates included concrete tankage, pumps and motors, piping, valves, fittings, structural steel, electrical and instrumentation, installation, and contingency (at 20 percent). Capital cost expressed as a function of hydraulic flow rate is graphed in Figure IX-18. Cost curves are shown for 25, 50, 75, and 100 percent recycle.

Annual Costs. Annual costs for wastewater recycle systems were assumed to include the following: (1) amortization calculated at 10 percent annual interest over 10 years for equipment (CRF = 0.16275) and 30 years for construction (CRF = 0.10608); (2) annual maintenance at three percent of total capital costs; (3) operator manhours calculated at $13.50 per hour (including benefits, insurance, etc.) for 20 hours per week; (4) energy computed at $0.03/KWH based on pumping horsepower at 75 percent efficiency and 75 feet total dynamic head, for which the following formulae apply:

$$\text{Horsepower} = \frac{\text{Wastewater flow (gpm)} \times 75 \text{ feet}}{0.75 \times 3960}$$

$$\text{Annual Energy Cost} = \text{Horsepower} \times 0.746 \text{ KW/HP} \times 24 \text{ hrs/day} \times 365 \text{ days/yr} \times \$0.03\text{/KWH};$$

and (5) additional monitoring at $10,000 per year. Total annual cost curves for 25, 50, 75, and 100 percent recycle systems are shown in Figure IX-19.

Evaporation Pond

A lined evaporation pond was costed for the only known discharging uranium mill (Mill 9405). The pond was estimated to require 380 acres of land area. Land costs were assumed to be $1,000/acre for this site alone. In addition, the pond was assumed to be located ten miles from the site for purposes of costing pump station and piping requirements. Piping distance was based upon statements of the company concerning the location of available land. Total capital and annual cost figures for this pond are documented in Table IX-10.

ACTIVATED CARBON ADSORPTION

Capital Costs

Systems have been costed for activated carbon adsorption of phenolic compounds. Figure IX-20 provides the equipment definition for these systems. Carbon contactor vessels are constructed of carbon steel. A backwash system is provided to remove suspended solids from the carbon contactors.

Carbon contactors are designed for 30-minute retention time and (100 lbs) of carbon for 0.23 kg (0.5 lbs) of phenol. A total phenol (4AAP) concentration of 0.4 mg/l was assumed for system sizing.

Total capital costs included equipment, installation, and contingency. Figure IX-21 graphically represents this capital cost as a function of hydraulic flow rate.

Annual Costs

Annual costs for activated carbon adsorption include capital cost amortization, maintenance, operation, energy, taxes and insurance, and off-site regeneration of carbon. Amortization was calculated at 10 percent annual interest rate over a 10 year life expectancy for equipment (CRF = 0.16275) and a 30 year life expectancy for construction (CRF = 0.10608). Annual maintenance costs were estimated at three percent of the initial capital investment. Operator manhours were established at 2,000 hours per year and costed at $13.50/hour including benefits. Activated carbon costs were based on a price of $0.50/lb.; energy was estimated at $0.03/KWH (3 cents); and taxes and insurance were estimated at two percent of the initial capital investment.

Figure IX-22 is a graphic display of the annual costs associated with activated carbon adsorption of phenolic compounds.

HYDROGEN PEROXIDE TREATMENT

Capital Costs

Cost estimates have been prepared for systems which oxidize phenolic compounds by the addition of hydrogen peroxide in the presence of ferrous sulfate catalyst. The design assumptions included the use of a 6:1:1 ratio of hydrogen peroxide:ferrous sulfate:phenol. The total phenol (4AAP) concentrations was assumed to be 0.4 mg/l.

Figure IX-23 is a schematic flow diagram of the system design. Oxidation basins are sized for five-minute retention time. Mixers assisted by an air buffing system are provided in the include air compressor, oxidation basins, mixers, clarifiers, sludge pumps, hydrogen peroxide storage tank, reagent pumps, instrumentation and localized piping as well as installation and contingency costs.

Figure IX-24 relates total capital costs for hydrogen peroxide oxidation systems to hydraulic flow rate.

Annual Costs

Annual costs associated with hydrogen peroxide oxidation systems have been estimated. Included in the estimates are capital cost amortization, maintenance, operation, energy, and chemical costs. Amortization was based on a 10 percent annual interest rate, a 10 year life expectancy for equipment (CRF = 0.16275) and a 30 year life expectancy for construction (CRF = 0.10608). Maintenance costs were estimated at three percent of the initial capital investment annually. Operator manhours were established as 2,000 hours per year at a rate of $13.50/hour including benefits. Energy costs were based on a rate of $0.03/KWH. Hydrogen peroxide was costed at $0.35/lb for a 70 percent concentration, sulfuric acid at $0.04/lb, and ferrous sulfate at $0.52/lb.

Taxes and insurance were estimated at two percent of the initial capital investment.

Figure IX-25 displays annual costs as a function of hydraulic flow rate for hydrogen peroxide treatment systems.

CHLORINE DIOXIDE TREATMENT

Capital Costs

Systems for the oxidation of phenolic compounds by chlorine dioxide addition have been estimated. Design assumptions include chlorine dioxide dosage of 6 mg/l, retention time of 10 minutes in the contact tank, and a 30-day reagent storage capacity. Figure IX-26 is a schematic flow diagram of the system including reagent storage tank, enclosure metering pump, contact tank, discharge pump, ejector, and filter.

Capital costs include equipment, construction, installation, localized piping and electrical work, instrumentation and contingencies. Figure IX-27 graphically displays capital costs for these systems as a function of hydraulic flow rate.

Annual Costs

Figure IX-28 shows the annual costs associated with chlorine dioxide oxidation systems as a function of hydraulic flow rate. Annual costs include capital cost amortization, maintenance, operation, energy, and chemical costs. Amortization was calculated at a 10 percent annual interest rate over a 10 year life expectancy for equipment (CRF = 0.16275) and a 30 year life expectancy for construction (CRF = 0.10608). Maintenance costs were estimated at three percent of the initial capital investment annually. Operator manhours were estimated at 2,000 hours per year at a rate of $13.50/hour including benefits. Energy costs were based on a rate of $0.03/KWH, (3 cents). Chlorine dioxide costs were estimated as $9.10/gal (5 percent conc.). Taxes and insurance were estimated to be two percent of the initial capital investement.

POTASSIUM PERMANGANATE OXIDATION

Capital Costs

Cost estimates have been prepared for the installation of systems which oxidize phenolic compounds by the use of potassium permanganate. The system definition is shown schematically in Figure IX-29.

Design assumptions include one hour retention time in the oxidation basins, neutral pH conditions, clarifier overflow rate permanganate dosage was estimated at 7:1 ratio of potassium permanganate to phenolics. A 0.4 mg/l concentration of total phenol (4AAP) was assumed.

Capital costs include equipment, installation, construction, localized piping and electrical work, instrumentation, and contingency costs. Figure IX-30 displays total capital costs as a function of hydraulic flow rate.

Annual Costs

Capital recovery was amortized over a 10-year period for equipment and a 30 year period for construction. A 10 percent annual interest rate was used (Equipment CRF = 0.16275, Construction CRF = 0.10608). Annual maintenance costs were assumed to be three percent of the capital investment. Operator manhours of 2,000 hours per year were costed at $13.50/hour including benefits, etc. Energy costs were estimated at $0.03/KWH. Potassium permanganate was costed at $0.59/lb (dry). Taxes and insurance were estimated to be two percent of the total capital cost. Figure IX-31 shows the total cost estimates for these systems.

MODULAR TREATMENT COSTS FOR THE ORE MINING AND DRESSING INDUSTRY

Tables IX-2 through IX-10 list unit treatment process costs for each facility studied. Costs are given in terms of a) Capital Cost ($1,000), b) Annual Costs ($1,000), and c) Cost: cents/ton of ore mined.

For purposes of these tabulations, the capital and annual cost curves of this section to which the additional costs of monitoring must be added where applicable were used.

NON-WATER QUALITY ISSUES

Solid Waste

Solid wastes generated during the ore mining and milling processes are currently being investigated by EPA for possible regulations under the Resource Conservation and Recovery Act (RCRA). Solid wastes from mining and milling operations include, but are not limited to: overburden, tailings, mine and mill wastewater treatment sludges, lean ore, etc. The EPA has sponsored several studies (References 4, 5, and 6) in response to Section 8002, p and f of RCRA. These studies have examined the sources and volumes of solid wastes generated, present disposal practices, and quality of leachate generated under test conditions. To date, leachate tests have been performed on approximately 370 ore mining and milling solid wastes. Solid wastes from all of the ore mining and dressing subcategories have been examined and only 11 samples (approximately three percent) were found which exceeded the RCRA EP (extraction procedure) criteria (References 4 and 5). The vast majority (approximately 97 percent) of the ore mining and milling solid wastes are not hazardous (EP toxic).

In addition, Section 7 of the Solid Waste Disposal Act Amendments of 1980 has exempted, under Subtitle C of the RCRA, solid waste from the extraction, beneficiation, and processing of ores and minerals. This exemption will remain in effect until at least six months after the administrator submits a study on the adverse environmental effects of solid wase from mining. This study is required to be submitted by 21 October 1983.

TABLE IX-1. COST COMPARISONS GENERATED ACCORDING TO TREATMENT PROCESS AND ORE CATEGORY

		Secondary Settling	Flocculant Addition	Ozonation	Alkaline Chlorination	Ion Exchange	Granular Media Filtration	pH Adjustment	RECYCLE				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Total Evaporation Ponds
									25%	50%	75%	100%					
Iron Ore	Mines	X	X				X										
	Mills	X	X				X										
Copper Ore	Mines	X	X	X	X		X	X									
	Mills	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	
Lead/Zinc Ores	Mines	X	X	X	X		X	X									
	Mills	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	
Gold/Silver Ores	Mines	X	X	X	X		X	X									
	Mills	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	
Aluminum Ore	Mines	X	X				X										
Ferroalloy Ores	Mines	X	X				X	X									
	Mills	X	X	X	X	X	X	X	X	X	X	X					
Mercury Ore	Mines																
	Mills																
Titanium Ore	Mine/Mill	X	X				X	X									
Uranium Ore	Mines	X	X			X	X	X									
	Mills	X						X				X					X

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 1 of 9

Type of Mine: Iron Ore

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	
1101-MN Mine	36,376	21.13	a. 340 b. 44.8 c. 0.12	110 160 0.44	-	-	-	2000 310 0.85	-	-	-	-	-	-	
1101-MN Mill	36,376	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1102-MN Mine	9,072	0.69	a. 84 b. 19.0 c. 0.21	65 30 0.33	-	-	-	150 47 0.52	-	-	-	-	-	-	
1102-MN Mill	9,072	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1103-MN Mine/Mill	4,409	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1104-MN Mine	1,808	1.05	a. 97 b. 20.2 c. 1.12	70 32 1.77	-	-	-	206 55 3.04	-	-	-	-	-	-	
1104-MN Mill	1,808	5.94	a. 187 b. 29 c. 1.60	90 58 3.21	-	-	-	808 140 7.74	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

p. 2 of 9

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-cula-tion	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
1105-MN	9,149	12.70	a. 250 b. 35 c. 0. 38	100 99 1.08	-	-	-	1400 220 2.40	-	-	-	-	-	-	
1105-MN Mill	9,149	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1106-MN Mine/Mill	44,092	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1107-MI Mine	5,842	3.53	a.150 b. 25.2 c. 0.43	82 45 0.77	-	-	-	525 100 1.71	-	-	-	-	-	-	
1107-MI Mill	5,842	2.69	a.136 b. 24 c. 0.41	80 40 0.68	-	-	-	430 85 1.45	-	-	-	-	-	-	
1108-MI Mine	9,700	4.17	a.161 b. 26.5 c. 0.27	85 50 0.52	-	-	-	605 110 1.13	-	-	-	-	-	-	
1108-MI Mill	9,700	8.71	a.225 b. 33 c. 0.34	95 75 0.77				1070 182 1.88							
1109-MI Mine	18,078	N.A.	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

p. 3 of 9

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
1109-MI Mill	18,078	5.94	a. 186 b. 29.2 c. 0.16	90 58 0.32	-	-	-	800 140 0.77	-	-	-	-	-	-	
1110-PA Mine	2,866	N.A.	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1110-PA Mill	2,866	1.71	a. 115 b. 22 c. 0.77	75 37 1.29	-	-	-	300 70 2.44	-	-	-	-	-	-	
1111-MN Mine	34,172	14.00	a. 280 b. 38.5 c. 0.11	100 110 0.32	-	-	-	1500 230 0.67	-	-	-	-	-	-	
1112-MN Mine	9,590	7.00	a. 210 b. 31.5 c. 0.33	91 65 0.68	-	-	-	900 150 1.56	-	-	-	-	-	-	
1112-MN Mill	9,590	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1113-MN Mine/mill	27,558	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1114-MO Mine	2,601	1.43	a. 107 b. 21.1 c. 0.81	72 35 1.34	-	-	-	253 62 2.38	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-cula-tion	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
1114-MO Mill	2,601	1.71	a. 115 b. 22 c. 0.85	75 37 1.42	-	-	-	300 70 2.69	-	-	-	-	-	-	
1115-MO Mine	2,425	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1115-MO Mill	2,425	4.17	a. 161 b. 26.5 c. 1.09	85 50 2.06	-	-	-	605 110 4.54	-	-	-	-	-	-	
1116-WI Mine/Mill	2,425	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1117-UT Mine/Mill	2,645	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1118-CA Mine/Mill	9,028	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-cula-tion	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust	Recycle				Process Control	
										25%	50%	75%	100%		
1119-WY Mine	4,850	0.45	a. 75 b. 18.2 c. 0.38	60 28 0.58	-	-	-	110 42 0.87	-	-	-	-	-	-	
1119-WY Mill	4,850	Minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1120-MI Mine	4,630	0.18	a. 56 b. 17 c. 0.37	54 27 0.58	-	-	-	50 33 0.71	-	-	-	-	-	-	
1120-MI Mill	4,630	Minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1121-MN Mine	1,194	5.94	a. 186 b. 29.2 c. 2.45	90 58 4.86	-	-	-	800 140 11.73	-	-	-	-	-	-	
1121-MN Mill	1,194	1.44	a. 108 b. 21.1 c. 1.77	72 35 2.93	-	-	-	254 63 5.28	-	-	-	-	-	-	
1122-MN Mine	8,157	17.80	a. 315 b. 42.3 c. 0.52	105 125 1.53	-	-	-	1750 280 3.43	-	-	-	-	-	-	
1122-MN Mill	8,157	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 6 of 9

Type of Mine: Iron Ore

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
1123-MN Mine	2,535	2.98	a. 140 b. 24.6 c. 0.97	81 43 1.70	-	-	-	480 91 3.59	-	-	-	-	-	-	
1123-MN Mill	2,535	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1124-MN Mine	11,905	2.98	a. 140 b. 24.6 c. 0.21	81 43 0.36	-	-	-	480 91 0.76	-	-	-	-	-	-	
1124-MI Mill	11,905	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1125-MN Mine/Mill	1,543	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1126-MN Mine/Mill	2,425	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1127-UT Mine/Mill	1,874	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1128-NM Mine/Mill	71.65	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

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Mine Code - Location Type	Ore Production (1000 tons/ year)	Water Dis- charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Floc- cula- tion	Ozon- ation	Alkal. Chlor- ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
1129-TX Mine	2,380	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1129-TX Mill	2,380	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1130-NY Mine/Mill	1,984	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1131-NY Mine	3,858	0.44	a. 74 b. 18.1 c. 0.47	60 28 0.73	-	-	-	108 41 1.06	-	-	-	-	-	-	
1131-NY Mill	3,858	16.36	a. 306 b. 38 c. 0.98	100 110 2.85	-	-	-	1650 250 6.48	-	-	-	-	-	-	
1132-WY Mine/Mill	1,433	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1133-MN Mine	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Operation closed
1134-MN Mine/Mill	1,433	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

p. 8 of 9

Mine Code - Location Type	Ore Production (1000 tons/ year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-cula-tion	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
1135-MN Mine/ Mill	1,212	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1136-MI Mine	301	120.46	a. 820 b. 96 c.31.89	140 860 285.71	-	-	-	6000 1060 352.16	-	-	-	-	-	-	
1137-CA Mine/ Mill	496	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1138-MN Mine/ Mill	9735	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1139-GA Mine	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Operation assumed closed
1140-MN Mine	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
1141-MN Mine	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
1142-MN Mine/ Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-2. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Iron Ore

p. 9 of 9

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
1143-MN Mine/Mill	2,648	1.06	a. 98 b. 20.2 c. 0.76	70 32 1.21				200 55 2.08							
1144-MI Mine	1,874	3.25	a. 148 b. 25.2 c. 1.34	82 43 2.30	-	-	-	500 95 5.07	-	-	-	-	-	-	
1145-NV Mine	115	0.63	a. 83 b. 18.8 c. 16.35	64 28 24.35	-	-	-	145 46 40.00	-	-	-	-	-	-	
1146-MN Mine	661	10.00	a. 240 b. 34.6 c. 5.23	98 82 12.41	-	-	-	1200 185 27.99	-	-	-	-	-	-	
1147-MN Mine	413	2.19	a. 126 b. 23 c. 5.57	79 37 8.96	-	-	-	360 79 19.13	-	-	-	-	-	-	
1148-MN Mill	1297	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
1149-MN Mill	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
			a. b. c.												

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined							Recycle									Remarks
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	
2101-NV Mine/Mill	7,932	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2102-AZ Mine/Mill	6,015	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2103-NM Mine/Mill	15,403	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2104-NM Mine/Mill	8,101	0.18	a. 61 b. 17 c. 0.21	56 27 0.33	32 27 0.33	48 27 0.33	850 250 3.09	50 33 0.41	26 23 0.28	7 16 0.20	9.5 16.5 0.20	13 17 0.21	17 18.5 0.23	120 67 0.83	140 91 1.12	108 83 1.02	160 99 1.22	–	
2107-AZ Mine/Mill	4,402	0	a. 61 b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Operation inactive
2108-AZ Mine/Mill	3,086	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2109-AZ Mine/Mill	3,729	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2110-AZ Mine	4,090	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Operation presently inactive

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: **Base and Precious Metals (Copper)**

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined																Remarks
										Recycle									
			Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	
2111-AZ Mine/Mill	1,631	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Inactive
2112-AZ Mine/Mill	670	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2113-AZ Mine/Mill	10,340	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2115-AZ Mine/Mill	1,555	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2116-AZ Mine/Mill	9,804	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2117-TN Mill	2,024	8.50	a. 220 b. 32 c. 1.58	95 75 3.71	540 157 7.76	230 237 11.71	11000 2760 136.36	1050 180 8.89	68 70 3.46	60 31 1.53	90 43 2.12	130 57 2.81	170 70 3.45	1650 805 39.77	330 195 9.63	285 185 9.14	1100 345 17.05		
2118-AZ Mine/Mill	18,357	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2119-AZ Mine/Mill	15,013	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	Remarks
2120-MT Mine	17,000	0.05	a. 58 b. 17 c. 0.10	45 26 0.15	20 24 0.14	42 25 0.15	–	18 28 0.16	23 22 0.13	–	–	–	–	–	–	–	–	–	
2120-MT Mill	17,000	9.50	a. 235 b. 34 c. 0.20	97 80 0.47	600 177 1.04	260 257 1.51	12000 3110 18.29	1150 190 1.12	70 75 0.44	65 33 0.19	100 46 0.27	145 62 0.36	180 75 0.44	1800 905 5.32	340 205 1.21	300 195 1.15	1200 360 2.12	–	
2121-MI complex	3,617	32	a. 400 b. 50 c. 1.38	120 210 5.81	1800 470 12.99	610 737 20.38	42000 13010 359.69	2500 400 11.06	125 190 5.25	165 70 1.94	270 125 3.46	380 170 4.70	485 230 6.36	5200 2705 74.79	580 400 11.06	540 390 10.78	3200 820 22.67	–	Already meeting BAT.
2122-UT Mill	35,500	8.50	a. 220 b. 32 c. 0.09	95 75 0.21	540 157 0.44	230 237 0.67	11000 2760 7.77	1050 180 0.51	68 70 0.20	60 31 0.08	90 43 0.12	130 57 0.16	170 70 0.20	1650 805 2.27	330 195 0.55	285 185 0.52	1100 345 0.97	–	
2123-AZ Mine/Mill	2,047	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2124-AZ Mine/Mill	6,710	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2125-AZ Mine	0	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Temporarily inactive
2126-NV Mine/Mill	8,000	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	Remarks
2130-NM Mine/Mill	NA	Minimal	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Mine dis-charges to mill – very small or zero discharge
2131-NV Mine	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2132-NV Mine/Mill	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2133-NV Mine/Mill	0	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Closed permanently
2134-ID Mine	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2134-ID Mill	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Partial recycle
2135-AZ Mine	8.27	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2136-AZ Mine	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Operation inactive

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

p. 5 of 7

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢ / ton or ore mined																Remarks
			Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	
										25%	50%	75%	100%						
2137-AZ Mine/Mill	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2138-AZ Mine/Mill	5,000	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2139-AZ Mine/Mill	32,494	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2140-AZ Mine/Mill	5,800	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2141-AZ Mine/Mill	5,300	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2142-AZ Mine	1,820	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2143-AZ Mine	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Suspected inactive
2144-AZ Mine	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

p. 6 of 7

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined							Recycle									Remarks
			Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	
2145-AZ Mine/Mill	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2146-AZ Mine/Mill	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2147-AZ Mine/Mill	19,600	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Temporarily inactive
2148-AZ Mine/Mill	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Probably inactive
2149-AZ Mine	NA	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
2150-UT Mine/Mill	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Under devel-opment – 0 discharge likely
2151-MI Mine/Mill	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Pilot-scale production
2152-NM Mill	0	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Temporarily inactive

TABLE IX-3. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Copper)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code · Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)		Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	Remarks
2154-AZ Mine	NA	NA	a.																	Under development or exploration
			b.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																	
			a.																	
			b.																	
			c.																	
			a.																	
			b.																	
			c.																	
			a.																	
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			b.																	
			c.																	
			a.																	
			b.																	
			c.																	
			a.																	
			b.																	
			c.																	

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	Remarks
										25%	50%	75%	100%						
3101 Mine/Mill	206	0.38	a. 70 b. 17.5 c. 8.50	60 27 13.11	50 35 16.99	54 32 15.53	1200 330 160.19	90 40 19.42	30 25 12.14	9.5 16 7.77	14 16.5 8.01	18 17.5 8.50	24 19.5 9.47	–	–	–	–	–	Closed
3102-MO Mine/Mill	1,634	5.94	a. 180 b. 29 c. 1.77	90 60 3.67	400 122 7.47	195 172 10.5	7500 2210 135.25	800 140 8.57	60 56 3.43	50 27 1.65	73 37 2.26	100 47 2.88	130 57 3.49	1260 600 36.72	290 165 10.10	250 160 9.79	870 270 16.52	–	
3103-MO Mine/Mill	1,072	2.58	a. 135 b. 24 c. 2.24	80 40 3.73	195 74 6.90	120 85 7.92	4000 1210 112.87	400 83 7.74	45 38 3.54	30 22 2.05	44 27 2.58	60 31 2.89	75 36 3.36	700 305 28.45	225 130 12.13	190 125 11.66	520 185 17.26	–	
3104-NY Mill	1,112	1.78	a. 115 b. 22 c. 1.98	75 37 3.33	145 59 5.31	92 65 5.85	3000 860 77.34	300 70 6.29	42 34 3.06	23 20 1.80	32 24 2.16	44 27 2.43	56 30 2.70	540 235 21.13	203 120 10.79	170 115 10.34	410 160 14.39	–	
3105-MO Mine	1,138	2.19	a. 124 b. 23 c. 2.02	78 38 3.34	180 66 5.80	110 75 6.59	–	340 75 6.59	43 36 3.16	–	–	–	–	–	–	–	–	–	
3106-PA Mine	383	28.53	a. 390 b. 48 c. 12.53	120 190 49.61	1680 422 110.18	585 667 174.15	–	2400 370 96.61	115 170 44.38	–	–	–	–	–	–	–	–	–	
3106-PA Mill	383	1.50	a. 110 b. 21.4 c. 5.59	75 35 9.14	130 54 14.10	88 59 15.40	2700 760 198.43	260 65 16.97	41 33 8.62	21 20 5.22	30 23 6.00	40 25 6.53	52 28 7.31	460 205 53.52	195 115 30.03	168 110 28.72	380 150 39.16	–	
3107-ID Complex	782	5.94	a. 180 b. 29 c. 3.71	90 60 7.67	400 122 15.60	195 172 21.99	7500 2210 282.61	800 140 17.90	60 56 7.16	50 27 3.45	73 37 4.73	100 47 6.01	130 57 7.29	1260 600 76.73	290 165 21.10	250 160 20.46	870 270 34.53	–	

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined							Recycle									Remarks
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	
3108-TN Mill	391	0.05	*a. 55	45	20	42	460	18	23	3.5	5	7	8	52	124	86	125		
			b. 16.5	24	24	25	150	28	22	15	16	16.5	17	47	85	75	89	–	
			c. 4.22	6.14	6.14	6.39	38.35	7.16	5.83	3.84	4.10	4.22	4.35	12.02	21.74	19.18	22.76		
3109-MO Mine/Mill	1,117	7.50	a. 210	95	490	220	9000	930	65	55	80	125	150	1500	315	275	1030		
			b. 32	67	147	217	2610	155	65	30	43	55	66	725	185	180	320	–	
			c. 2.86	6.00	13.16	19.43	233.66	13.88	5.82	2.09	3.85	4.92	5.91	64.91	16.56	16.11	28.65		
3110-NY Mill	103	0.58	a. 80	68	64	62	1600	140	33	13	17	24	30	250	160	134	240		
			b. 18.6	28	36	36	450	46	27	17	18	19	22	115	99	94	120	–	
			c. 18.06	27.18	34.95	34.95	436.89	44.66	26.21	16.50	17.48	18.45	21.36	111.65	96.12	91.26	116.50		
3111-TE Mine	100	0.95	a. 94	70	92	72		190	36										
			b. 19.9	32	43	47	–	54	28	–	–	–	–	–	–	–	–	–	
			c. 19.90	32.00	43.0	47.00		54.00	28.00										
3112-NM Mine	135	0.66	a. 84	65	69	64		145	34										
			b. 18.9	30	38	39	–	47	26	–	–	–	–	–	–	–	–	–	
			c. 14.00	22.22	28.15	28.89		34.81	19.26										
3113-CO Mine	203	1.69	a. 112	75	140	92	2700	280	42	23	32	45	55						
			b. 22.0	34	57	62	810	67	33	20	23	27	30	–	–	–	–	–	
			c. 10.84	16.75	28.08	30.54	399.01	33.00	16.26	9.85	11.33	13.30	14.78						
3113-CO Mill	203	1.40	a. 105	75	120	85	2500	250	40	20	28	39	49	450	190	165	360		
			b. 21.0	34	53	58	730	61	32	19	22	25	28	195	115	108	150	–	
			c. 10.34	16.75	26.11	28.57	359.61	30.05	15.76	9.36	10.84	12.32	13.79	96.06	56.65	53.20	73.89		
3114-ID Mine/Mill	68	0.42	a. 74	61	52	56	1400	95	31	11	15	19	24	200	155	126	205		
			b. 18.0	27	33	33	360	41	26	16	16.5	17.5	20	97	95	92	115	–	
			c. 26.47	39.71	48.53	48.53	529.41	60.29	38.23	23.53	24.26	25.74	29.41	142.65	139.71	135.29	169.12		

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined																Remarks
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	
										25%	50%	75%	100%						
3115 Mine/Mill	372	4.7	a. 160 b. 26 c. 6.99	90 51 13.71	320 98 26.34	165 147 39.52	6000 1610 432.80	640 120 32.26	53 50 13.44	43 25 6.72	60 32 8.60	84 39 10.48	110 45 12.10	–	–	–	–	–	Closed. Annual production based on 250 working days
3116-CO Mine	198	0.87	a. 91 b. 19.6 c. 9.90	69 32 16.16	84 42 21.21	70 45 22.73	–	180 51 25.76	36 28 14.14	–	–	–	–	–	–	–	–	–	
3118-VA Mine	596	1.80	a. 118 b. 22.1 c. 3.71	75 35 5.87	146 60 10.07	96 67 11.24	2900 810 135.91	310 72 12.08	42 34 5.70	23 20 3.36	32 23 3.86	45 26 4.36	60 30 5.03	–	–	–	–	–	Ore production includes Mine 3117
3118-VA Mine	596	2.60	a. 135 b. 23.7 c. 3.98	80 40 6.71	196 75 12.58	120 86 14.42	3900 1110 186.24	400 88 14.77	46 38 6.38	30 22 3.69	42 26 4.36	58 30 5.03	73 35 5.87	–	–	–	–	–	Ore production includes Mine 3117
3118-VA Mill	596	0.01	a. 55 b. 16.5 c. 2.77	35 25 4.19	15 21 3.52	42 25 4.19	330 120 20.13	10 17 2.85	18 22 3.69	1.7 15 2.52	2.5 16 2.68	3.1 16.5 2.77	4.2 17 2.85	20 40 6.71	120 85 14.26	70 70 11.74	115 85 14.26	–	
3118-VA Mill	596	0.20	a. 60 b. 16.8 c. 2.82	55 26 4.36	33 27 4.53	48 28 4.70	880 250 41.95	54 33 5.54	28 23 3.86	6.9 16 2.68	9.5 16.5 2.77	12 17 2.85	16 18 3.02	125 70 11.74	140 90 15.10	110 82 13.76	165 100 16.78	–	
3118-VA Mine/Mill	596	14.00	a. 280 b. 38 c. 6.38	100 110 18.46	860 242 40.60	350 357 59.89	16000 4510 756.71	1500 230 38.60	85 100 16.78	87 40 6.71	130 59 9.90	170 80 13.42	240 100 16.78	2500 1255 210.57	400 245 41.11	350 240 40.27	1650 465 78.02	–	
3119-MO Mine/Mill	647	1.78	a. 115 b. 22.0 c. 3.40	75 35 5.41	145 59 9.12	92 65 10.04	2900 810 125.19	310 72 11.13	42 34 5.26	23 20 3.09	32 23 3.55	45 26 4.02	60 30 4.64	540 235 36.32	203 120 18.55	170 115 17.78	410 160 24.73	–	

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined							Recycle									Remarks
			Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	
3120-ID Mine/Mill	174	1.24	a. 102 b. 20.7 c. 11.90	71 33 18.97	110 49 28.16	80 52 29.88	2300 660 373.56	230 60 34.48	39 31 17.82	19 19 10.92	26 21 12.07	37 24 13.79	47 27 15.52	410 180 103.45	190 113 64.94	160 104 59.77	340 145 83.33	–	
3121-ID Mine	283	1.24	a. 102 b. 20.7 c. 7.31	71 33 11.66	110 49 17.31	80 52 18.37	–	230 60 21.20	39 31 10.95	–	–	–	–	–	–	–	–	–	
3121-ID Mill	283	1.58	a. 111 b. 21.6 c. 7.63	75 34 12.01	136 55 19.43	88 60 21.20	2900 780 275.62	270 66 23.32	41 34 12.01	22 20 7.07	30 23 8.13	42 26 9.19	52 29 10.25	500 215 75.97	200 120 42.40	165 110 38.87	390 150 53.00	–	
3122-MO Mine	1,111	6.90	a. 205 b. 30.9 c. 2.78	95 65 5.85	460 139 12.51	210 197 17.73	–	900 160 14.40	63 62 5.58	–	–	–	–	–	–	–	–	–	
3123-MO Mine	1,774	9.60	a. 230 b. 34.0 c. 1.92	98 80 4.51	600 177 9.98	260 257 14.49	–	1165 191 10.77	71 76 4.28	–	–	–	–	–	–	–	–	–	
3124-NJ Mine	205	0.25	a. 64 b. 17.3 c. 8.44	57 27 13.17	37 29 14.15	50 30 14.63	–	65 35 17.07	28 23 11.22	–	–	–	–	–	–	–	–	–	
3125-NY Mine	24	0.37	a. 72 b. 17.8 c.74.17	60 28 116.67	47 32 133.33	54 31 129.17	–	90 38 158.33	30 24 100.00	–	–	–	–	–	–	–	–	–	
3127-TN Mine	721	1.45	a. 108 b. 21.2 c. 2.94	75 34 4.72	125 53 7.35	85 56 7.76	–	255 63 8.74	40 31 4.29	–	–	–	–	–	–	–	–	–	

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	Remarks
										25%	50%	75%	100%						
3128-TN Mine	526	1.45	a. 108 b. 21.2 c. 4.03	75 34 6.46	125 53 10.07	85 56 10.64	–	256 63 11.98	40 31 5.89	–	–	–	–	–	–	–	–	–	
3130-UT Mine	NA	8.50	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Inactive
3131-WI Mine	NA	2.00	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
3132-WI Mine	NA	1.16	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
3133-WI Mine	0	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
3133-WI Mill	0	0.76	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
3134-WA Mine	301	0.71	a. 86 b. 19.0 c. 6.31	66 30 9.97	73 40 13.29	65 41 13.62	–	165 50 16.61	35 28 9.30	–	–	–	–	–	–	–	–	–	
3135-WA Mine/Mill	*	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	*Production included with Mine 3134

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu- lation	Ozona- tion	Alkal. Chlorin- ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang- anate	Process Control	Remarks
3136-NV Mine/Mill	126	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Pilot scale operation
3137-AZ Mine/Mill	93.2	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
3138-CO Mine	98.2	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Discharges twice yearly
3139-IL Mill	NA	0.87	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
3140-NM Mill	144	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
3141-TN Mine	0	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Operation closed
3141-TN Mill	0	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
3142-UT Mine/Mill	216	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	Remarks
3143-CO Mine	60	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
3143-CO Mill	60	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4103-CO Mine/Mill	0	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Operation closed
UKA-WI Mine	NA	1.00	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
UKB-WI Mine	NA	1.90	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Presently inactive
UKC-TN Mine/Mill	NA	2.00	a. 120 b. 22.5 c. –	79 37.5 –	155 62 –	100 71 –	3200 910 –	330 75 –	43 35 –	26 21 –	37 24 –	50 28 –	63 32 –	580 256 –	210 125 –	176 117 –	440 165 –	–	
UKD-TN Mine	NA	4.49	a. 163 b. 27 c. –	89 51 –	310 103 –	170 137 –	–	630 110 –	55 50 –	–	–	–	–	–	–	–	–	–	Under-ground Mine
UKD-TN Mill	NA	1.06	a. 95 b. 20 c. –	70 32 –	98 44 –	74 48 –	2200 610 –	208 58 –	38 29 –	17 18 –	24 20 –	32 22 –	41 26 –	375 165 –	180 105 –	155 100 –	310 135 –	–	Under-ground Mine

TABLE IX-4. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Lead-Zinc)

p. 8 of 8

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined																Remarks
										Recycle									
			Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	
7 (100)-TN Mine/Mill	NA	0	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
7 (102)-CO Mine/Mill	NA	NA	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-5. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Gold-Silver)

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	Remarks
4101-NV Mine/Mill	820	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4102-CO Mine	179	1.00	a. 96	70	95	74	2100	200	36										
			b. 20	32	43	47	600	55	28	–	–	–	–	–	–	–	–	–	
			c. 11.17	17.88	24.02	26.26	335.20	30.73	15.64										
4102-CO Mill	287	0.36	a. 72	60	50	53	1200	90	30	9.5	15	18	23	180	152	122	195		
			b. 17.8	28	31	32	330	38	24	16	17	18	20	88	95	91	108	–	
			c. 6.20	9.76	10.80	11.15	114.98	13.24	8.36	5.57	5.92	6.27	6.97	30.66	33.10	31.70	37.63		
4104-WA Mine/Mill	55	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4105-SD Mine	1,560	3.04	a. 145	82	220	130		480	48										
			b. 24.7	43	79	97	–	92	44	–	–	–	–	–	–	–	–	–	
			c. 1.58	2.76	5.06	6.22		5.90	2.82										
4115-UT Mine/Mill	145	Minimal	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4116-AZ Mine/Mill	NA	NA	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4117-NV Mine/Mill	80	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																

TABLE IX-5. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: **Base and Precious Metals (Gold-Silver)**

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton or ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Floccu-lation	Ozona-tion	Alkal. Chlorin-ation	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permang-anate	Process Control	Remarks
										25%	50%	75%	100%						
4118-NV Mine/Mill	NA	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4119-NV Mine/Mill	NA	Minimal	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4120-NV Mine/Mill	NA	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4121-NV Mine/Mill	0	0	a.																Presently inactive
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4122-NV Mine/Mill	0	0	a.																Presently inactive
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4123-NV Mine	NA	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4124-NM Mine	0	0	a.																Temporarily inactive
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																
4126-AK Mine/Mill	NA	0	a.																
			b. –	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
			c.																

TABLE IX-5. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Gold-Silver)

p. 3 of 4

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined																Remarks
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	
										25%	50%	75%	100%						
4127-AK Mine/Mill	612 (m^3)	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4128-NV Mine/Mill	730	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4129-CO Mine	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Under exploration
4129-CO Mill	0.18-0.36	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Under exploration
4130-NV Mine	0	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Inactive
4131-NV Mine/Mill	2,004	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4401-ID Mine	181	0.21	a. 63 b. 17.1 c. 9.45	55 28 15.47	34 28 15.47	48 28 15.47	–	57 34 18.78	27 23 12.71	–	–	–	–	–	–	–	–	–	
4401 and 4406-ID Mills	181 (4401) 106 (4406)	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	Wastewater in combined tailings pond

TABLE IX-5. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Base and Precious Metals (Gold-Silver)

p. 4 of 4

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton or ore mined																Remarks
										Recycle									
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Activated Carbon	Hydrogen Peroxide	Chlorine Dioxide	Potassium Permanganate	Process Control	
4402-CO Mine	74.4	0.78	a. 88 b. 19.4 c. 26.08	68 30 40.32	78 40 53.76	69 43 53.76	–	166 50 67.20	35 27 36.29	–	–	–	–	–	–	–	–	–	
4403-ID Mill	198	0.83	a. 90 b. 19.5 c. 9.85	69 30.5 15.40	85 41 20.71	–	1900 510 257.58	175 53 26.77	34 28 14.14	16 17 8.59	22 19 9.60	29 21 10.61	38 24 12.12	320 136 68.18	180 105 53.03	145 98 49.50	282 127 64.14	–	
4404-CO Mine	407	1.00	a. 96 b. 20 c. 4.91	70 32 7.86	95 44 10.81	74 47 11.55	–	200 55 13.51	36 28 6.88	–	–	–	–	–	–	–	–	–	
4406-ID Mine	108	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–		–	–	–	–	
4407-ID Mine	684	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4408-MT Mine	74	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4409-ID Mine	NA	NA	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
4410-ID Mine	84	0	a. b. – c.	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	

TABLE IX-6. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 1 of 1

Type of Mine: Aluminum Ore

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
5101-AR Mine	1,200	1.90	a. 120 b. 22.4 c. 1.87	77 37 3.08	-	-	-	340 75 6.25	-	-	-	-	-	-	
5102-AR Mine	872	3.67	a. 152 b. 25.8 c. 2.96	85 47 5.39	-	-	-	546 102 11.70	-	-	-	-	-	-	
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 1 of 8

Type of Mine: Ferroalloy

TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton of ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	REMARKS
6101-NM Mill	6,283	2.90	a. 140 b. 24 c. 0.38	82 43 0.68	215 77 1.23	130 94.0 1.50	4500 1260 20.05	480 92 1.46	47 40 0.64	32 23 0.36	47 28 0.44	62 33 0.52	80 38 0.60	-	
6102-CO Mill	15,430	2.90	a. 140 b. 24 c. 0.16	82 43 0.28	215 77 0.50	130 94.0 0.61	4500 1260 8.17	480 92 0.60	47 40 0.26	32 23 0.15	47 28 0.18	62 33 0.21	80 38 0.24	-	
6103-CO Mine	2,425	2.87	a. 140 b. 24 c. 0.99	82 43 1.77	-	-	-	480 90 3.71	47 40 1.65	-	-	-	-	-	
6104-CA Mine	705	8.71	a. 230 b. 32 c. 4.54	97 77 10.92	-	-	-	1071 182 25.82	69 71 10.07	-	-	-	-	-	
6105-NY Mine/Mill	11	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6106-OR Mine/Smelter	1,322	Minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6107-AZ Mine	361	5.54	a. 180 b. 28 c. 7.76	90 56 15.51	-	-	-	750 140 38.78	60 55 15.24	-	-	-	-	-	
6108-NV Mine/Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 2 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined								Recycle				REMARKS
			Second. Settling	Floc-culation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
6109-CA Mine/Mill	16	Minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6110-ID Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Exploratory operations underway
6111-AK Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
6112-NC Mine/Mill	ca. 330	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Temporarily inactive-under exploration
6113-NM Mine	50	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6114-NV Mine	0	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Exploratory operations underway
6115-CO Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6116-SC Mine/Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 3 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-culation	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
6117-NV Mine/Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6118-MN Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6119-CO Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Exploratory operations
6120-UT Mine	0	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6121-NV Mine	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6122-CA Mine	ca. 11	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6123-ID Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6124-CA Mine	ca.33	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 4 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
6125-CA Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6126-ID Mine	NA	Intermittent	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Presently inactive
6127-ID Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Exploration underway
6128-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6129-NV Mine	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6130-NV Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6131-CA Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
6132-UT Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 5 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
6133-MT Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6134-ID Mine	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6135-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6136-UT Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6137-UT Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6138-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6139-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6140-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 6 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/ year)	Water Dis- charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc- cula- tion	Ozon- ation	Alkal. Chlor- ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	
6141-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6142-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6143-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6144-CA Mine	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6145-CA Mine/Mill	0.11	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6146-CA Mine/Mill	0.5	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6147-CA Mine	0.11	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6148-NV Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 7 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-culation	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	
6149-ID Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
6150-ID Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
6151-MT Mill	0.002	Minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6152-OR Mine	1	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6153-NV Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6154-MT Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6155-NV Mill	0.028	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6156-UT Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

TABLE IX-7. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 8 of 8

Type of Mine: Ferroalloy

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
6157-NV Mill	0.055 (conc.)	0	a. - b. c.	-	-	-	-	-	-	-	-	-	-	-	
6158-CA Mill	0.496	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Production based on mill capacity
6159-NN Mill	0.110 (conc.)	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6160-TX Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6161-NM Mill	8 (conc.)	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6162-SC Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
6163-CA Mill	0.055 (conc.)	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
			a. b. c.												

TABLE IX-8. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 1 of 1

Type of Mine: Mercury

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-culation	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
9201-CA Mine/Mill	30	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9202-NV Mine/Mill	17.5	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												
			a. b. c.												

TABLE IX-9. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 1 of 1

Type of Mine: Titanium

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-culation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
9905-NY Mine	464	0.70	a. 84 b. 19.0 c. 4.09	66 30 6.47	72 40 8.62	-	-	160 47 10.13	33 27 5.82	-	-	-	-	-	
9906-FL Mill	7260	6.84	a. 200 b. 30.5 c. 0.42	90 66 0.91	455 137 1.89		8100 2210 30.44	900 145 2.00	63 63 0.87	50 28 0.39	78 40 0.55	110 50 0.69	140 58 0.80	-	
9907-FL Mill	7260	1.63	a. 114 b. 21.8 c. 0.30	75 34 0.47	138 56 0.77		2800 800 11.02	270 66 0.91	40 33 0.45	22 20 0.28	30 23 0.32	40 27 0.37	52 29 0.40	-	
9908-FL Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
9909-FL Mine/Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	"
9910-NJ Mill	6600	3.77	a. 155 b. 26 c. 0.39	85 46 0.70	270 92 1.39		5300 1410 21.36	560 100 1.52	50 45 0.68	36 23 0.35	52 30 0.45	70 37 0.56	90 42 0.64	-	
9911-NJ Mine/Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Inactive
			a. b. c.												

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

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Type of Mine: Uranium, Radium, Vanadium

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
9401-NM Mine	750	0.85	a. 90 b. 19.5 c. 2.60	68 30 4.00	-	-	1900 510 68.00	180 52 6.93	35 28 3.73	-	-	-	-	-	
9401-NM Mill	1,270	(1.86)**	a. 120 b. 22 c. 1.72	-	-	-	-	-	-	-	-	-	60 30 2.36	-	No point discharge
9402-NM Mine 35,36	1,125	3.00	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9402-NM Mine 17,22 24,30,33		0.00	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	0 Discharge Ref. p. III-71
9402-NM Mill	2409	(1.94)**	a. 120 b. 22 c. 0.91	-	-	-	-	-	42 35 1.45	-	-	-	60 30 1.25	-	Mill receives ore from other mines
9403-UT Mill	274*	(0.41)**	a. 74 b. 18 c. 6.57	-	-	-	-	-	30 25 9.12	-	-	-	25 20 7.30	-	No point discharge
9404-NM Mill	2,490	(1.38)**	a. 110 b. 21 c. 0.84	-	-	-	-	-	39 31 1.24	-	-	-	47 27.5 1.10	-	"
9405-CO⁺ Mill	439*	1.00	a. 96 b. 20 c. 4.55	-	-	-	-	-	37 29 6.61	-	-	-	40 26 5.92	-	Single discharging uranium mill

* indicates production obtained from mill capacity - assume 365 working days/year.

( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

⁺See also page 5 for off site evaporation pond design.

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 2 of 5

Type of Mine: Uranium, Radium, Vanadium

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust	25%	50%	75%	100%	Process Control	
9407-WY Mill	724	(1.11)**	a. 95 b. 20 c. 2.76	-	-	-	-	-	37 30 4.14	-	-	-	-	-	No point discharge: recycle
9409-WY Mine	500	0.50	a. 75 b. 18.5 c. 3.70	-	-	-	-	-	32 26 5.20	-	-	-	-	-	
9409-WY Mill	1,086	(1.22)**	a. 100 b. 21 c. 1.93	-	-	-	-	-	38 30 2.76	-	-	-	44 27 2.48	-	No point discharge
9410-WY Mine	0	2.30	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	No production at present
9411-WY Mill	357	(0.39)**	a. 74 b. 18 c. 5.04	-	-	-	-	-	30 25 7.00	-	-	-	-	-	50% recycle: no discharge
9413-WY Mine	595	0.36	a. 74 b. 18 c. 3.03	-	-	-	-	-	30 25 4.20	-	-	-	-	-	
9413-WY Mill	603	(2.04)**	a. 120 b. 22.5 c. 3.73	-	-	-	-	-	43 36 5.97	-	-	-	-	-	No point discharge
9419-TX Mill	1,046	4.65	a. 172 b. 27.6 c. 2.64	-	-	-	-	-	56 49 4.68	-	-	-	-	-	" Recycle and lining

* indicates production obtained from mill capacity - assume 365 working days/year.

( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Uranium, Radium, Vanadium

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Mine Code - Location Type	Ore Production (1000 tons/ year)	Water Dis- charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined							Recycle					REMARKS
			Second. Settling	Floc- cula- tion	Ozon- ation	Alkal. Chlor- ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	25%	50%	75%	100%	Process Control	
9422-CO Mill	165	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Lined evapor. pond: no discharge
9423-WA Mill	183	(0.14)**	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	No point discharge
9425-WY Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9427-WY Mill	346*	(0.33)**	a. 70 b. 17.5 c. 5.06	-	-	-	-	-	29 24 6.94	-	-	-	23 19 5.49	-	No point discharge
9430-UT Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9437-NM Mine	96	5.03	a. 180 b. 28 c. 29.17	90 54 56.25	-	-	6500 1710 1781.25	700 130	58 52 54.17	-	-	-	-	-	
9442-WY Mill	563*	(0.28)**	a. 66 b. 17.4 c. 3.09	-	-	-	-	-	-	-	-	-	-	-	No point discharge
9443-NM Mine	NA	1.45	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	Under development

* Indicates production obtained from mill capacity - assume 365 working days/year.
( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

Type of Mine: Uranium, Radium
Vanadium

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TREATMENT TECHNOLOGIES AND COSTS:
a. Capital Cost ($1000)
b. Annual Cost ($1000)
c. Cost: ¢/ton of ore mined

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	REMARKS
9445-NM Mill	540	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9446-NM Mill	607*	(0.63)**	a. 80 b. 18.8 c. 3.10	-	-	-	-	-	35 27 4.45	-	-	-	-	-	No point discharge
9447-UT Mine	262	minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9447-UT Mill	273*	(0.31)**	a. 70 b. 17.5 c. 6.41	-	-	-	-	-	-	-	-	-	-	-	No point discharge
9449-WY Mine	750	9.8	a. 240 b. 34 c. 4.53	98 80 10.67	-	-	-	1200 190 25.33	71 78 10.4	-	-	-	-	-	
9449-WY Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9450-WY Mill	483*	(0.55)**	a. 80 b. 18.5 c. 3.83	-	-	-	-	-	34 26 5.38	-	-	-	29 22 4.55	-	No point discharge
9452-NM Mine	500	1.80	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

* indicates production obtained from mill capacity - assume 365 working days/year.
( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 4 of 5

Type of Mine: Uranium, Radium Vanadium

Mine Code - Location Type	Ore Production (1000 tons/ year)	Water Dis-charged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Floc-cula-tion	Ozon-ation	Alkal. Chlor-ination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle 25%	Recycle 50%	Recycle 75%	Recycle 100%	Process Control	
9445-NM Mill	540	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9446-NM Mill	607*	(0.63)**	a. 80 b. 18.8 c. 3.10	-	-	-	-	-	35 27 4.45	-	-	-	-	-	No point discharge
9447-UT Mine	262	minimal	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9447-UT Mill	273*	(0.31)**	a. 70 b. 17.5 c. 6.41	-	-	-	-	-	-	-	-	-	-	-	No point discharge
9449-WY Mine	750	9.8	a. 240 b. 34 c. 4.53	98 80 10.67	-	-	-	1200 190 25.33	71 78 10.4	-	-	-	-	-	
9449-WY Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9450-WY Mill	483*	(0.55)**	a. 80 b. 18.5 c. 3.83	-	-	-	-	-	34 26 5.38	-	-	-	29 22 4.55	-	No point discharge
9452-NM Mine	500	1.80	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	

* indicates production obtained from mill capacity - assume 365 working days/year.

( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

TABLE IX-10. COST COMPARISON FOR VARIOUS TYPES OF TREATMENT TECHNOLOGIES AND SUBSEQUENT COST PER TON OF ORE MINED (1 tn = 2000 lbs)

p. 5 of 5

Type of Mine: Uranium, Radium, Vanadium

Mine Code - Location Type	Ore Production (1000 tons/year)	Water Discharged (MGD)	TREATMENT TECHNOLOGIES AND COSTS: a. Capital Cost ($1000) b. Annual Cost ($1000) c. Cost: ¢/ton of ore mined												REMARKS
			Second. Settling	Flocculation	Ozonation	Alkal. Chlorination	Ion Exchan.	Mixed Media Filtr.	pH Adjust.	Recycle				Process Control	
										25%	50%	75%	100%		
9452-NM Mill	1,448*	(1.04)**	a. 95 b. 20 c. 1.38	-	-	-	-	-	37 30 2.07	-	-	-	40 26 1.79	-	
9456-WA Mill	764*	(0.63)**	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
New Mine "A" - NM Mine	0	0.87	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	No production at present
9460-WY Mine	300	0.87	a. 90 b. 19.5 c. 6.50	68 30 10.00	-	-	1900 510 170.00	180 52 17.33	36 28 9.33	-	-	-	-	-	
9460-WY Mill	NA	0	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9463-TX Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
Union Carbide-Rifle Mill	NA	NA	a. b. - c.	-	-	-	-	-	-	-	-	-	-	-	
9405-CO Mill	439	1.00	a. 23,900 b. 2,550 c. 580	-	-	-	-	-	-	-	-	-	-	-	Evaporation pond design

* indicates production obtained from mill capacity - assume 365 working days/year.

( )** for 0 discharge mills, flow indicated = volume discharged to treatment or recycle system.

TABLE IX-11. SUMMARY OF RESULTS FOR RCRA, EP ACETIC ACID LEACHATE TEST (all values as mg/l)

Waste Type by Segment	Arsenic Range	Barium Range	Cadmium Range	Chromium Range	Lead Range	Mercury Range	Selenium Range	Silver Range
Iron Mining and Milling (1101, 1105, 1109, 1120)								
Mine Waste Rock	<0.0005-0.161	0.025-0.715	<0.008-0.016	<0.001-0.003	<0.084	<0.0005-0.001	0.001-0.009	<0.002-0.008
Low Grade Ore	<0.0005-0.005	0.105-0.51	<0.008	<0.001-0.009	<0.084	<0.0005-0.001	0.003-0.009	<0.002-<0.003
Fresh Tailings	<0.0005-0.010	0.13-0.39	<0.008-0.021	<0.001-0.076	<0.084	<0.0005-0.001	0.003-0.015	<0.002-0.01
Tailing Ponds - Settled Solids	<0.0005-0.550	0.02-0.41	<0.008	<0.001-0.013	0.084-0.112	<0.0005-0.001	0.001-0.021	<0.002
Copper Mining and Milling (2101, 2104, 2118, 2119, 2120, 2121, 2122, 2126, 2139, 2147, 2164)								
Mine Waste Rock	0.002-0.050	0.058-0.62	<0.008-0.17	<0.001-<0.04	<0.06-0.840	<0.0005-0.002	0.0005-0.079	<0.002-0.024
Low Grade Ore	<0.002-0.0155	0.032-0.11	<0.008-0.071	<0.001-0.052	<0.08-0.084	<0.0005-0.001	0.006-0.056	<0.002-0.010
Fresh Tailings	0.006-0.055	0.04-2.8	<0.008-0.022	<0.001-0.057	<0.084-0.840	<0.0005-0.001	0.006 0.104	<0.002-0.012
Tailing Ponds - Settled Solids	0.0026-0.065	<0.001-2.0	<0.008-0.039	<0.001-0.110	<0.06-0.084	<0.0005-0.002	<0.001-0.105	<0.002-0.021

TABLE IX-11. SUMMARY OF RESULTS FOR RCRA, EP ACETIC ACID LEACHATE TEST (all values as mg/l) (continued)

Waste Type by Segment	Arsenic Range	Barium Range	Cadmium Range	Chromium Range	Lead Range	Mercury Range	Selenium Range	Silver Range
Lead/Zinc Mining and Milling (3104, 3106, 3107, 3110, 3113, 3122, 3123, 3126)								
Mine Waste Rock	<0.0041-0.047	0.052-0.270	0.039-0.650	0.004-0.180	<0.084-23.0	<0.0005-0.018	<0.001-0.032	<0.002-0.014
Fresh Tailings	<0.002-<0.023	0.051-0.665	<0.008-0.17	0.003-0.090	<0.084-16.0	<0.0005-0.041	0.006-0.050	<0.002-0.251
Tailing Ponds - Settled Solids	<0.002-0.043	0.016-1.68	<0.008-0.36	0.007-0.140	<0.06-8.8	<0.0001-0.102	<0.001-0.106	<0.002-0.180
Mine Water Ponds - Settled Solids	<0.0044-0.008	0.47-1.5	0.013-0.040	0.057-0.060	<0.084-0.100	0.023-0.027	0.007-0.039	<0.002-0.007
Gold/Silver Mining and Milling (4101, 4105, 4119, 4121, 4402, 4407)								
Mine Waste Rock	<0.0005-0.027	0.095-2.90	<0.003-0.098	<0.001-0.009	<0.06-11.0	<0.0005-0.001	<0.001-0.041	<0.002-0.034
Low Grade Ore	<0.002-0.103	0.160-3.25	<0.008-0.087	<0.001-0.087	<0.060-4.600	<0.0005	0.002-0.050	0.002-0.049
Fresh Tailings	0.004-0.017	0.009-1.73	<0.008-0.170	<0.001-0.120	<0.060-17.0	<0.0005-0.009	0.009-0.672	0.002-0.130
Tailing Ponds - Settled Solids	0.007-0.369	<0.001-1.90	<0.008-0.3	<0.001-0.074	<0.060 100.0	<0.0005-0.033	0.013 0.194	<0.002-0.065
Aluminum Mining (5101)								
Mine Water Ponds - Settled Solids	<0.025	0.15-1.19	<0.005-0.01	<0.05	0.14-0.23	<0.0003	<0.025	<0.001
Mine Waste Rock	<0.025	0.34	<0.005	<0.05	0.13	<0.0003	<0.025	<0.001

TABLE IX-11. SUMMARY OF RESULTS FOR RCRA, EP ACETIC ACID LEACHATE TEST (all values as mg/l) (continued)

Waste Type by Segment	Arsenic Range	Barium Range	Cadmium Range	Chromium Range	Lead Range	Mercury Range	Selenium Range	Silver Range
Molybdenum Mining and Milling (6101, 6102, 6103)								
Low Grade Ore	<0.005-<0.006	0.058-0.155	<0.008	0.001-<0.002	<0.084	<0.0005	0.004-0.018	<0.002
Mine Waste Rock	<0.005	0.08-0.19	<0.008	<0.001-0.004	<0.084	<0.0005	0.002-0.010	<0.002
Fresh Tailings	<0.0005-0.019	0.14-0.27	<0.008	<0.001-0.01	<0.084	<0.0005	0.002-0.043	<0.002-0.004
Tailing Ponds - Settled Solids	<0.005-0.017	0.09-0.2	<0.008	<0.001-0.018	<0.084-0.19	<0.0005-0.0018	0.003-0.023	<0.002-0.015
Wastewater Treatment Sludge	0.026	0.039	0.064	0.21	<0.084	<0.0005	0.055	0.03
Mine Water Pond - Settled Solids	0.048	0.74	<0.008	0.12	<0.084	<0.0005	0.006	0.011
Tungsten Mining and Milling (6104, 6105)								
Mine Waste Rock	<0.001-<0.002	0.22-0.4	0.015-0.02	<0.001-0.07	<0.05-<0.084	<0.0005	0.0199-0.052	<0.002-<0.01
Tailing Pond - Settled Solids	0.0218-0.075	0.395-0.59	0.017-0.027	0.0085-0.032	<0.06-<0.084	<0.0005-0.0005	0.0448-0.173	<0.002-<0.01
Dry Tailings	<0.001-0.02	0.2-0.4	<0.01-0.01	<0.04	<0.05	0.0001-0.0004	0.041-0.046	<0.01
Mine Water Pond Settled Solids	<0.002-<0.003	0.31-0.38	0.011-0.015	<0.001-0.002	<0.084	<0.0005-<0.0018	0.0048-0.0212	<0.002-0.002

TABLE IX-11. SUMMARY OF RESULTS FOR RCRA, EP ACETIC ACID LEACHATE TEST (all values as mg/l) (continued)

Waste Type by Segment	Arsenic Range	Barium Range	Cadmium Range	Chromium Range	Lead Range	Mercury Range	Selenium Range	Silver Range
Vanadium Mining and Milling (6107)								
Mine Waste Rock	<0.001	0.13	<0.01	<0.04	<0.05	<0.0001	<0.001	<0.002
Mine Water Pond - Settled Solids	<0.001	0.15	0.04	<0.04	<0.05	<0.0001	0.008	0.004
Tailing Pond - Settled Solids	<0.001	1.31-1.69	<0.01	<0.04	<0.05	<0.0001-0.0002	<0.001-0.004	<0.002-0.046
Mill Wastewater Ponds - Settled Solids	<0.001-0.54	0.03-0.3	<0.01-0.37	<0.04-1.9	<0.05	<0.0001-0.0036	0.001-0.03	0.008-0.25
Nickel Mining, Milling and Smelting (6106)								
Mine Waste Rock	0.02	0.1	<0.01	<0.04	<0.05	<0.0001	0.001	<0.002
Low Grade Ore	<0.001	<0.1	<0.01	<0.04	<0.05	<0.0001	<0.001	0.032
Mine/Smelter Wastewater Settled Solids	0.001	0.2	<0.01	<0.04	<0.05	0.0001	0.001	0.004
Mercury Mining and Milling (9202)								
Fresh Tailings	0.17	0.76	<0.005	<0.05	0.14	0.0019	<0.015	<0.001
Tailing Ponds - Settled Solids	0.26	0.76	<0.005	<0.05	0.11	0.041	<0.015	<0.001
Mine Waste Rock	0.1	0.76	0.01	0.06	<0.1	0.14	<0.015	<0.001

TABLE IX-11. SUMMARY OF RESULTS FOR RCRA, EP ACETIC ACID LEACHATE TEST (all values as mg/l) (continued)

Waste Type by Segment	Arsenic Range	Barium Range	Cadmium Range	Chromium Range	Lead Range	Mercury Range	Selenium Range	Silver Range
Uranium Mining (9402, 9403, 9404, 9405, 9408, 9409, 9411, 9412, 9423, 9447, 9451, 9455, 9460)								
Mine Waste Rock	<0.0005-0.031	0.001-1.29	<0.008-0.040	<0.001-0.056	<0.06-0.100	<0.0005-0.046	<0.0005-0.085	<0.002-0.06
Low Grade Ore	<0.0005-0.023	0.059-0.83	<0.008-0.040	0.004-<0.02	<0.06-<0.084	<0.0005-<0.014	<0.0005-0.154	0.005-0.018
Mine Water Ponds - Settled Solids	<0.0005-0.057	0.26-48.0	<0.008-0.040	<0.001-0.060	<0.06-0.100	<0.0005-0.009	<0.0005-0.073	<0.002-0.026
Titanium Dredge Mining and Milling (9906)								
Fresh Tailings	<0.001	<0.1	<0.01	<0.04	<0.05	0.0003	0.007	0.03
Mine Water Pond - Settled Solids	0.02	<0.1	<0.01	<0.04	<0.05	0.0004	0.008	0.016
Mill Wastewater Pond - Settled Solids	0.001	<0.1	<0.01	<0.04	<0.05	0.0005	0.006	0.015
Antimony Mining and Milling (9901)								
Fresh Tailings	0.21	1.85	0.02	0.06	0.14	<0.0003	<0.015	<0.001
Tailing Pond - Settled Solids	0.25	1.0	<0.005	<0.05	0.14	0.0003	<0.015	<0.001

Figure IX-1. SECONDARY SETTLING POND/LAGOON – TYPICAL LAYOUT

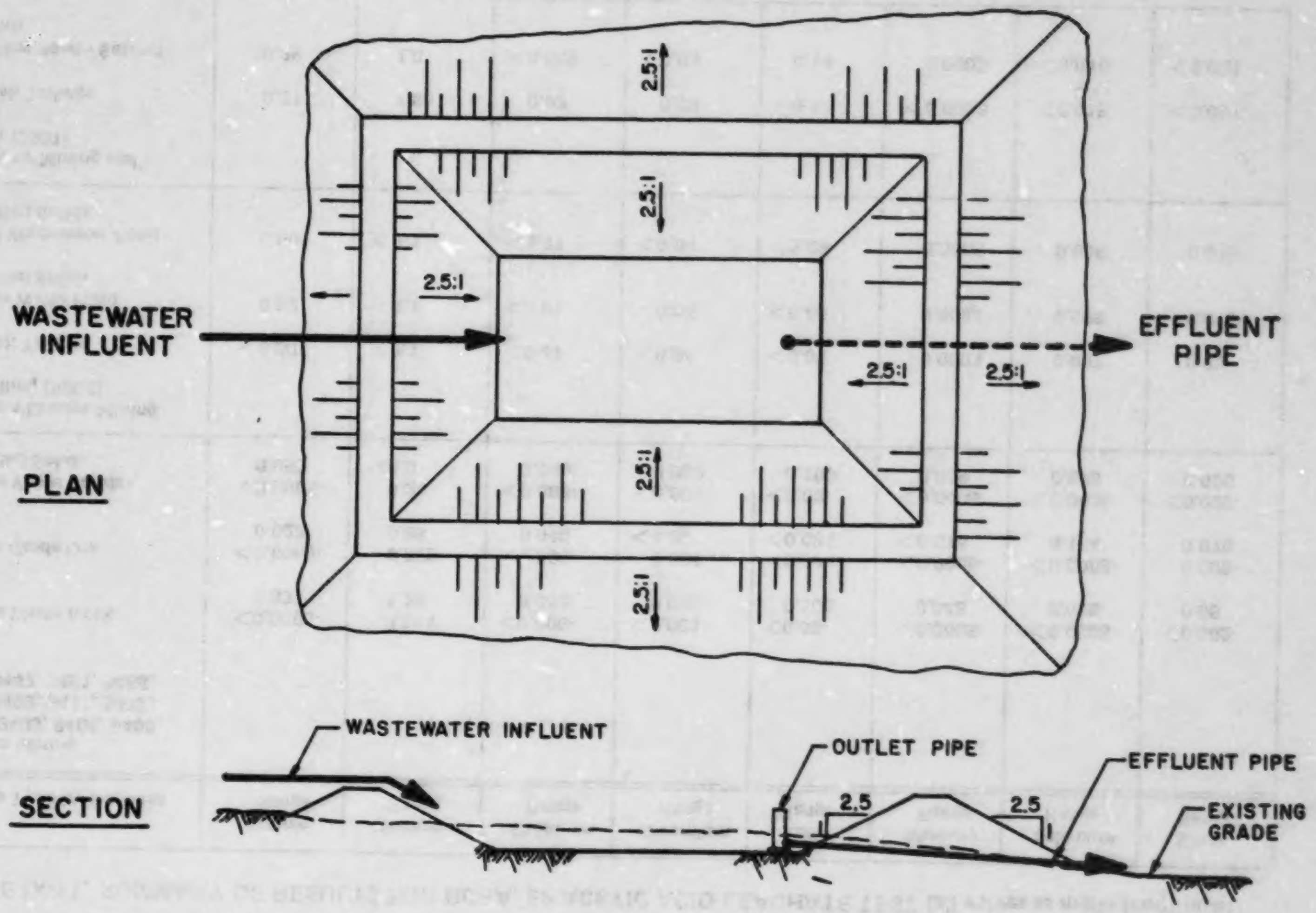
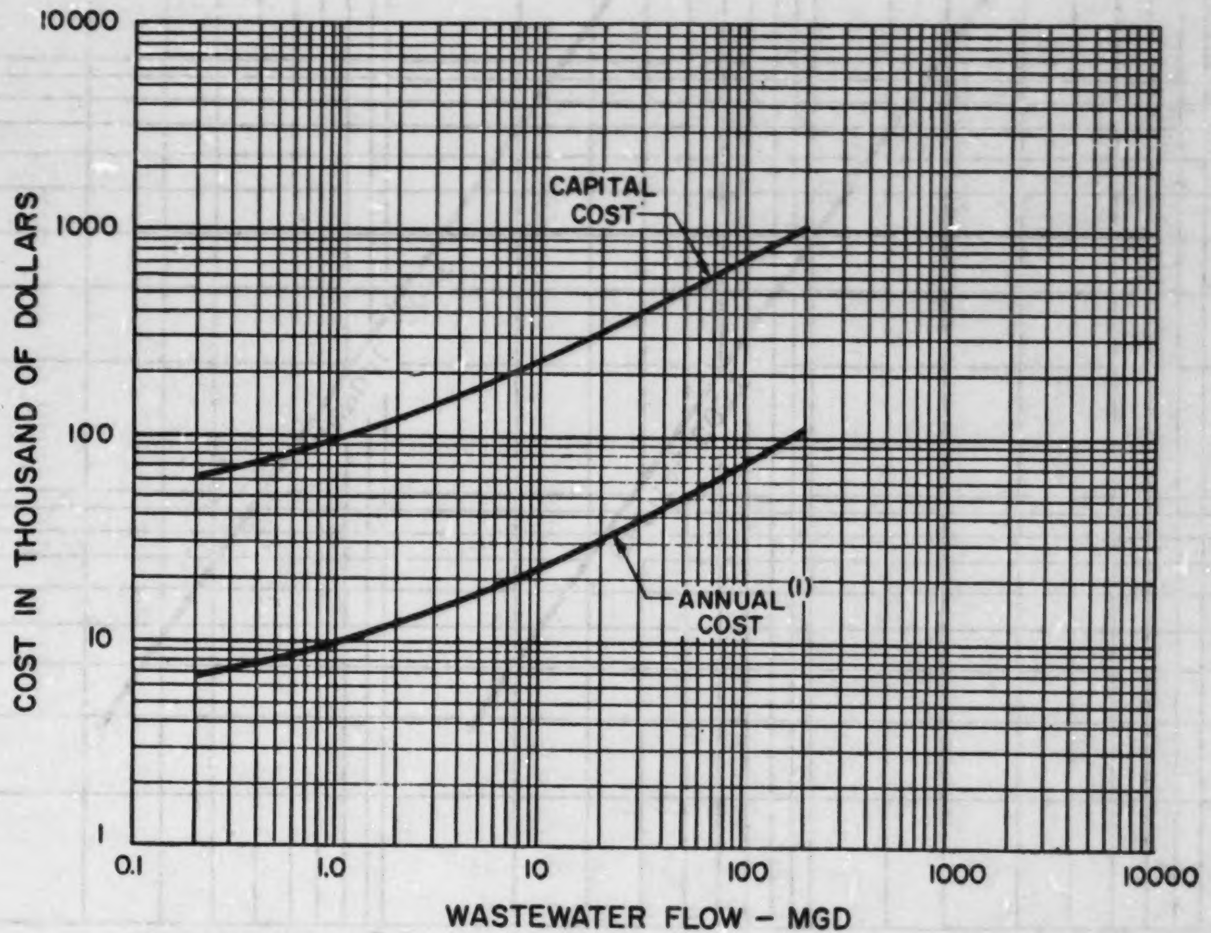

WASTEWATER INFLUENT
EFFLUENT PIPE
2.5:1
2.5:1
2.5:1
2.5:1
2.5:1
2.5:1
2.5:1
PLAN
WASTEWATER INFLUENT
OUTLET PIPE
2.5
2.5
1
1
EFFLUENT PIPE
EXISTING GRADE
SECTION

Figure IX-2. ORE MINING WASTEWATER TREATMENT SECONDARY SETTLING POND/LAGOON COST CURVES



(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED

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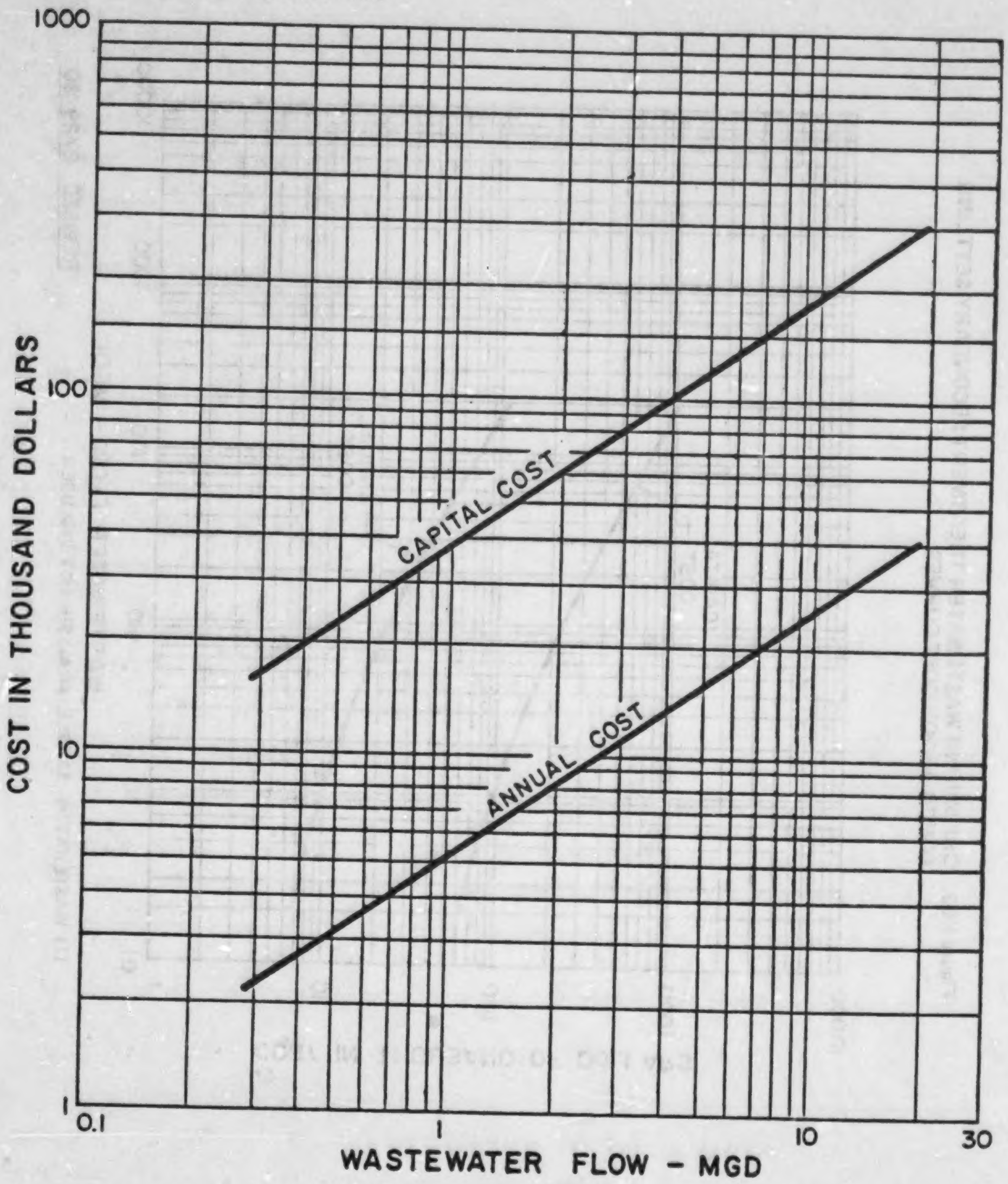
Figure IX-3. ORE MINE WASTEWATER TREATMENT SETTLING PONDS - LINING COST CURVES



NOTE: POND LINER USED FOR MINE 9405 ONLY

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Figure IX-4. FLOCCULANT (POLYELECTROLYTE) PREPARATION AND FEED-FLOW SCHEMATIC

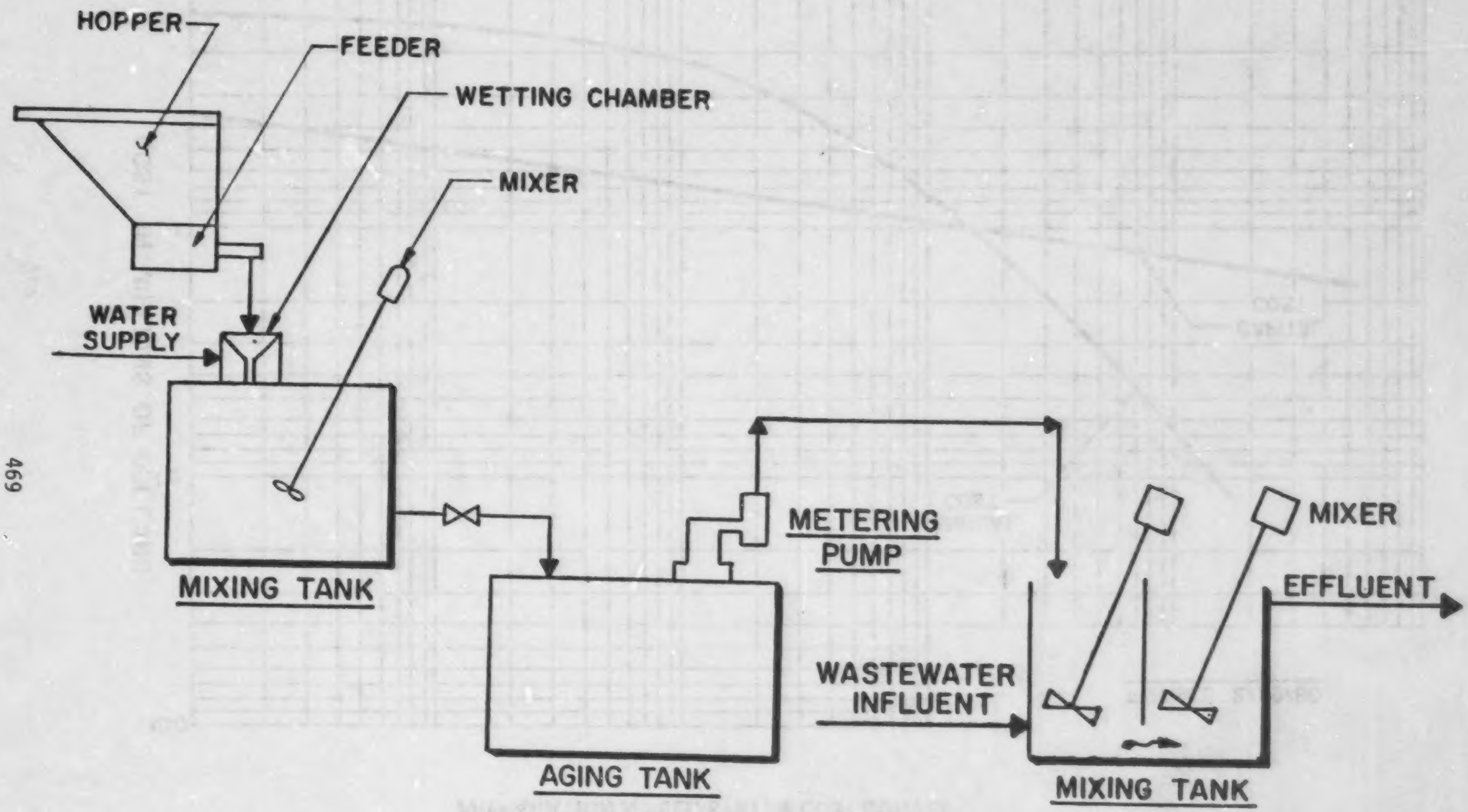
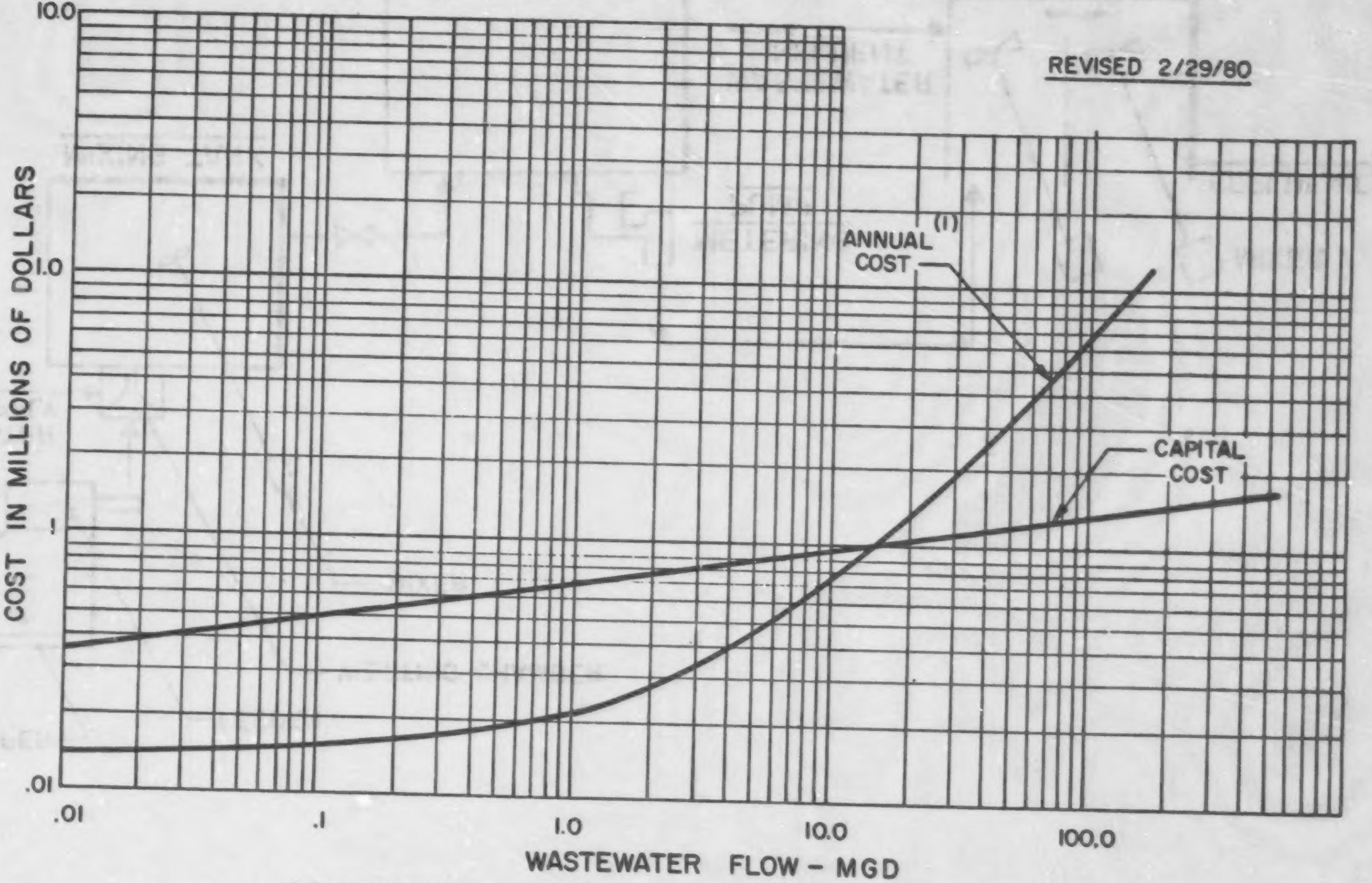

HOPPER
FEEDER
WETTING CHAMBER
MIXER
WATER SUPPLY
MIXING TANK
AGING TANK
METERING PUMP
WASTEWATER INFLUENT
MIXER
EFFLUENT
MIXING TANK

Figure IX-5. ORE MINE WASTEWATER TREATMENT FLOCCULANT (POLYELECTROLYTE) PREPARATION & FEED SYSTEM COST CURVES



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(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED

Figure IX-6. OZONE GENERATION AND FEED FLOW SCHEMATIC

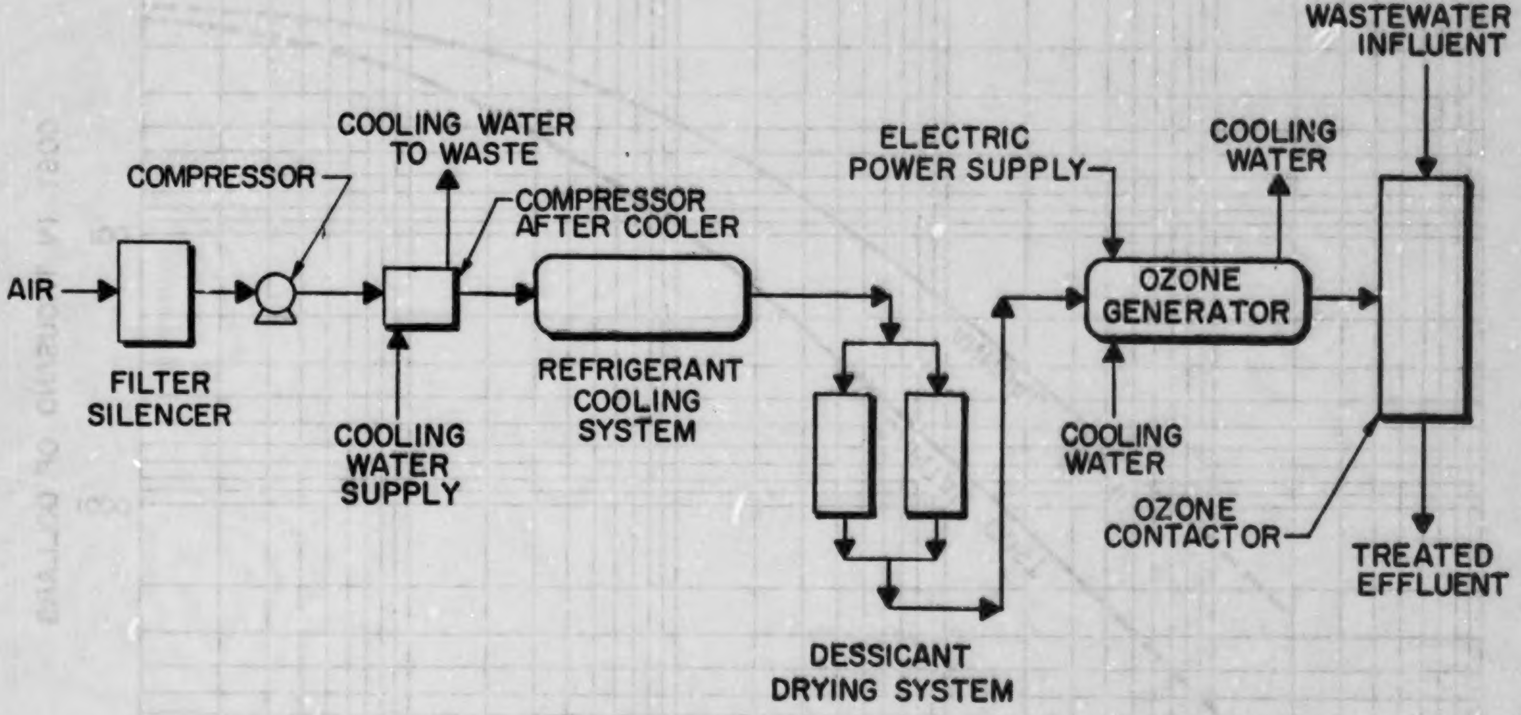
WASTEWATER INFLUENT
AIR
FILTER SILENCER
COMPRESSOR
COOLING WATER TO WASTE
COMPRESSOR AFTER COOLER
COOLING WATER SUPPLY
REFRIGERANT COOLING SYSTEM
DESSICANT DRYING SYSTEM
ELECTRIC POWER SUPPLY
COOLING WATER
OZONE GENERATOR
COOLING WATER
OZONE CONTACTOR
TREATED EFFLUENT

Figure IX-7. ORE MINE WASTEWATER TREATMENT OZONE GENERATION & FEED SYSTEM COST CURVES

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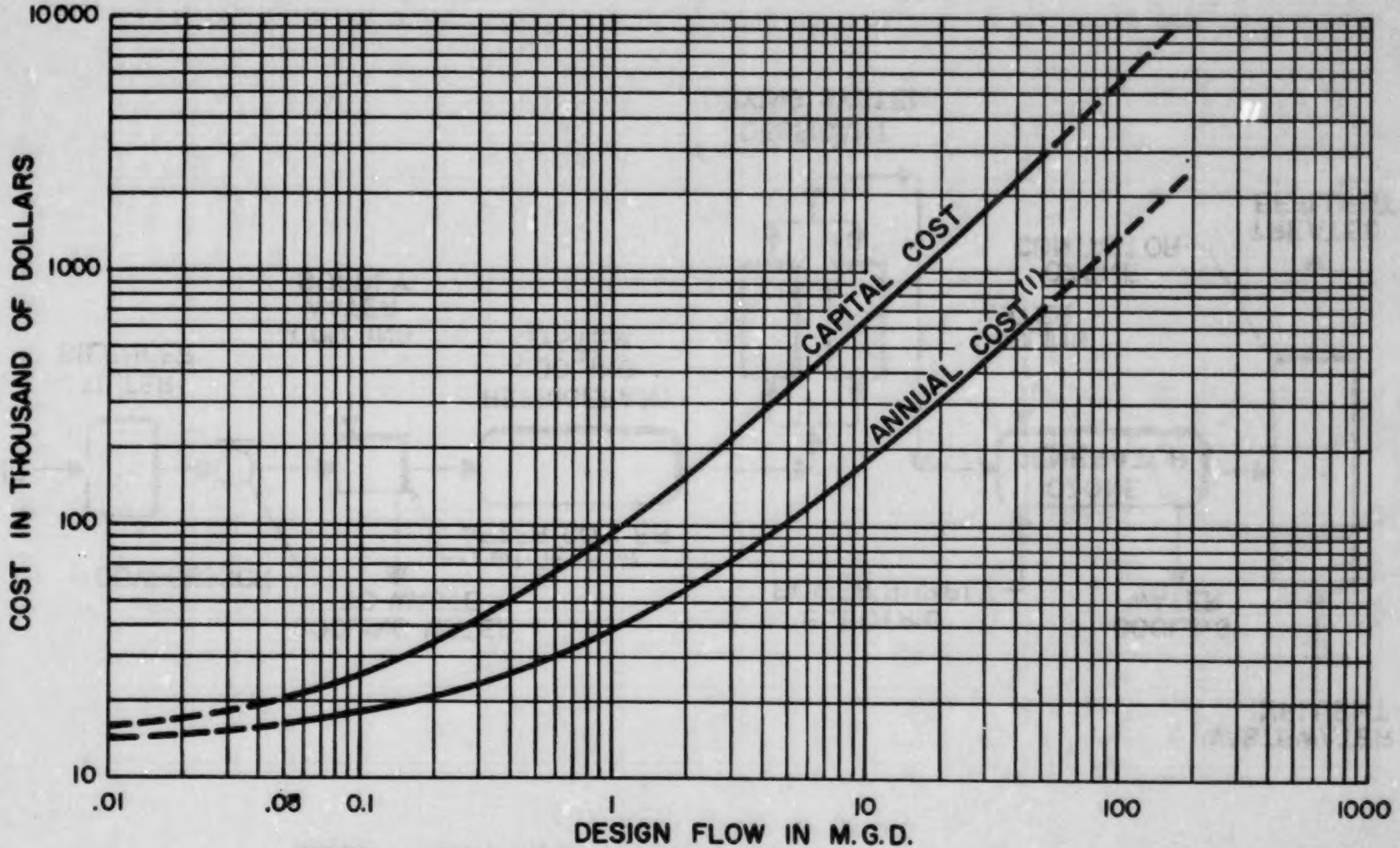


NOTES: 1) OZONE DOSAGE ASSUMED AT 5 mg/l.
2)(1)WASTEWATER SAMPLE ANALYSIS NOT INCLUDED.

Figure IX-8. ALKALINE-CHLORINATION FLOW SCHEMATIC

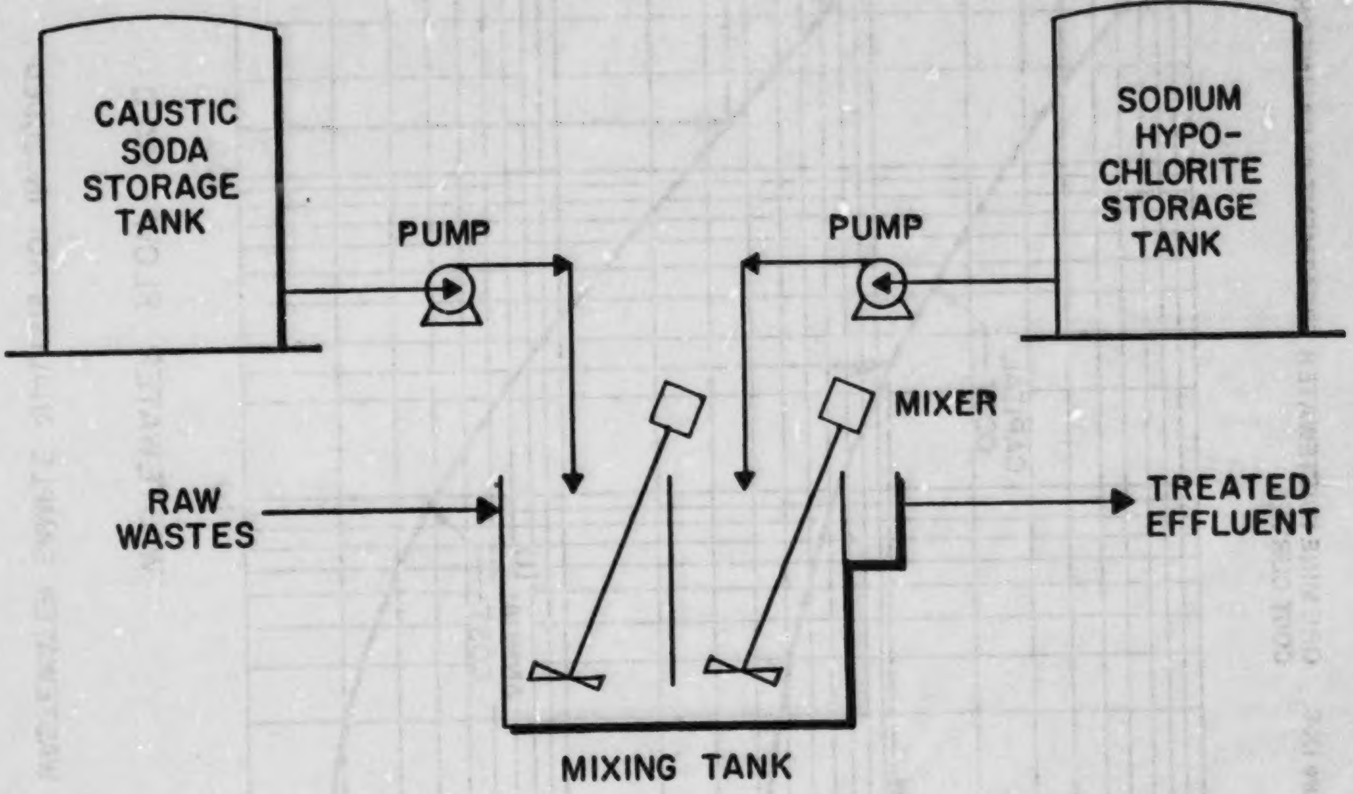
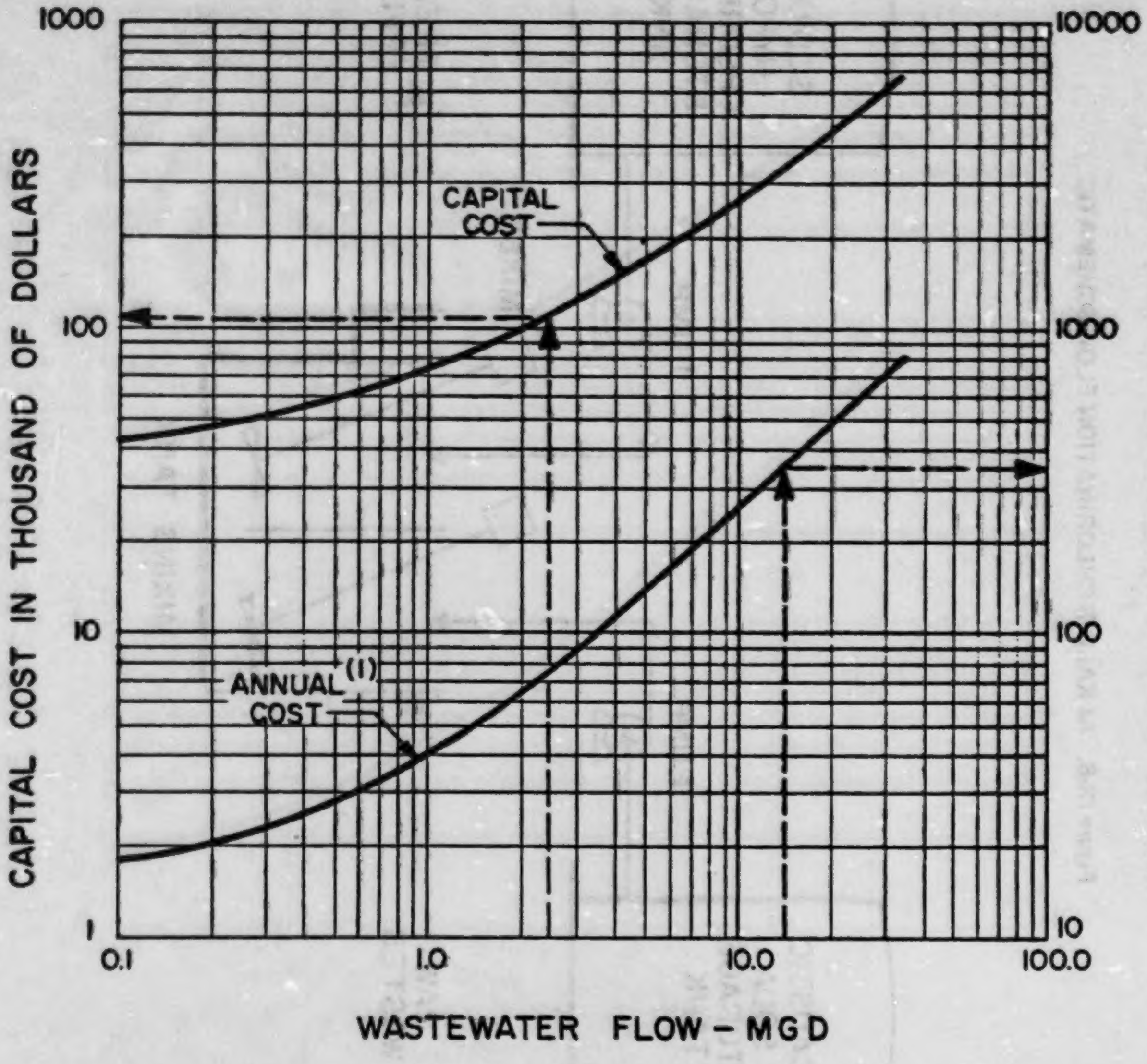

CAUSTIC SODA STORAGE TANK
PUMP
PUMP
SODIUM HYPO-CHLORITE STORAGE TANK
MIXER
RAW WASTES
TREATED EFFLUENT
MIXING TANK

Figure IX-9. ORE MINE WASTEWATER TREATMENT ALKALINE CHLORINATION COST CURVES



(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED

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Figure IX-10. ION EXCHANGE FLOW SCHEMATIC

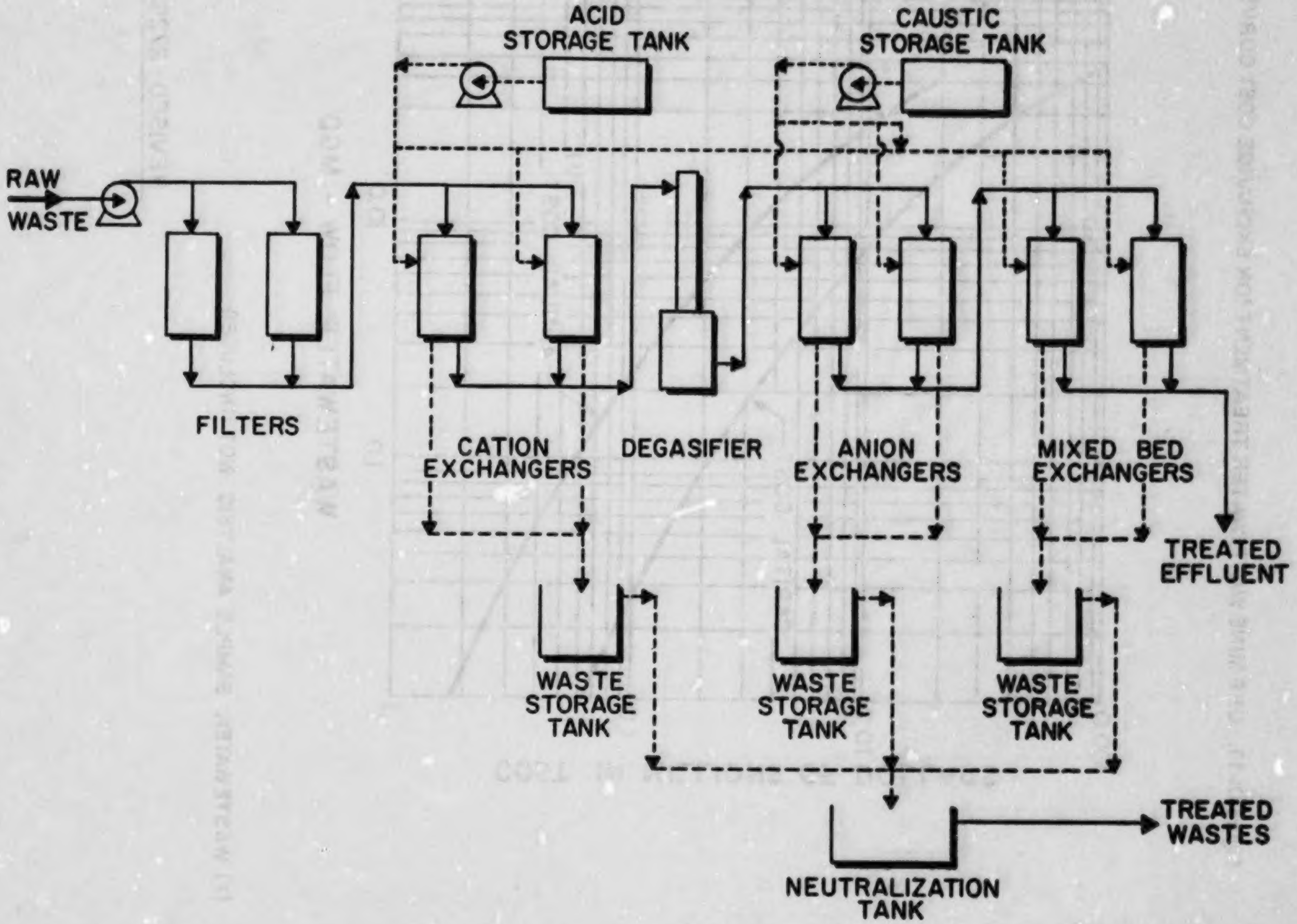
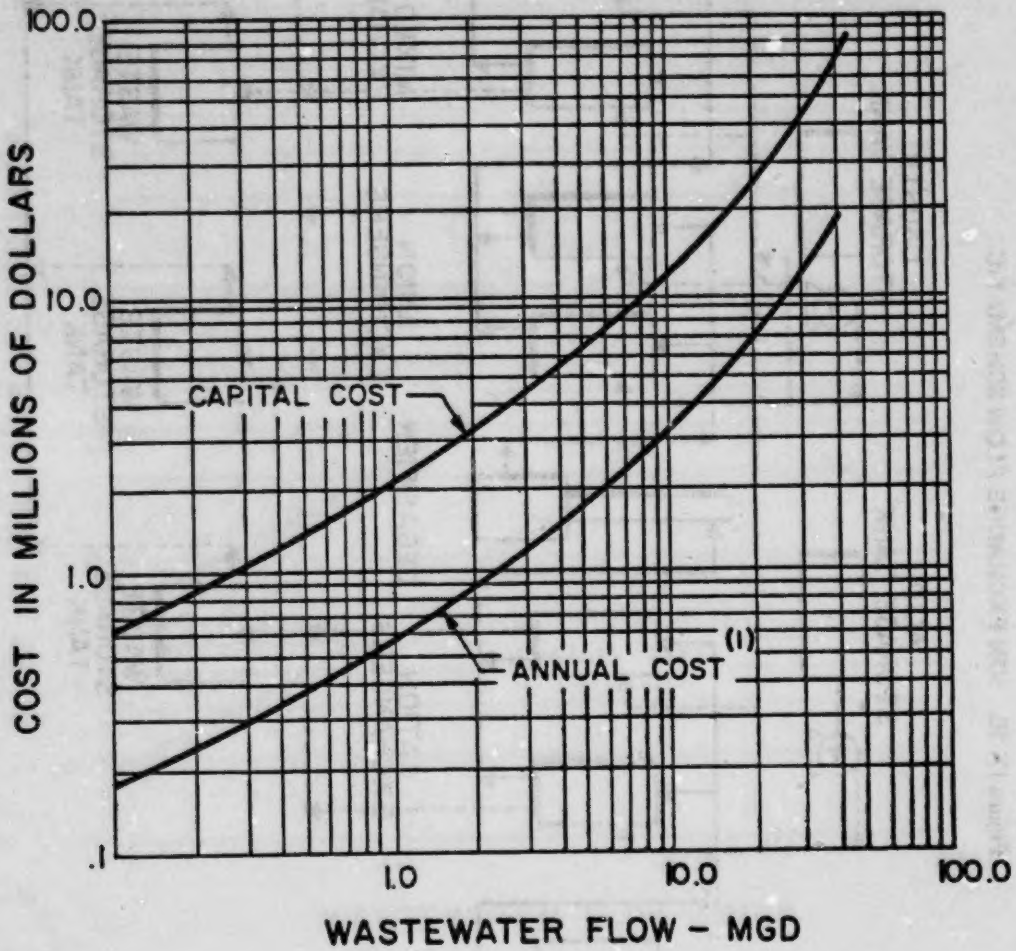

ACID STORAGE TANK
CAUSTIC STORAGE TANK
RAW WASTE
FILTERS
CATION EXCHANGERS
DEGASIFIER
ANION EXCHANGERS
MIXED BED EXCHANGERS
TREATED EFFLUENT
WASTE STORAGE TANK
WASTE STORAGE TANK
WASTE STORAGE TANK
NEUTRALIZATION TANK
TREATED WASTES

Figure IX-11. ORE MINE WASTEWATER TREATMENT ION EXCHANGE COST CURVES



(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED

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Figure IX-12. GRANULAR MEDIA FILTRATION PROCESS FLOW SCHEMATIC

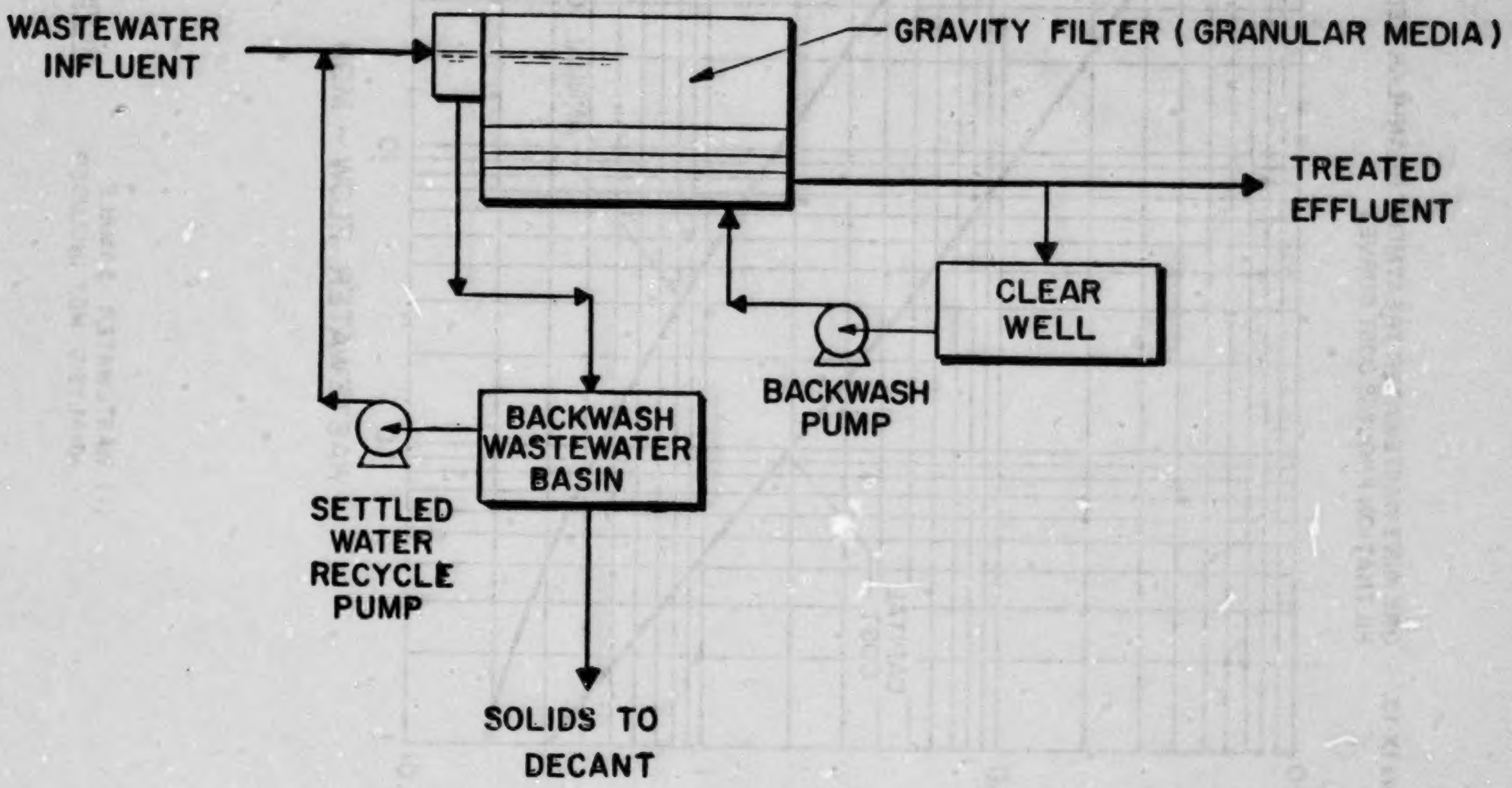
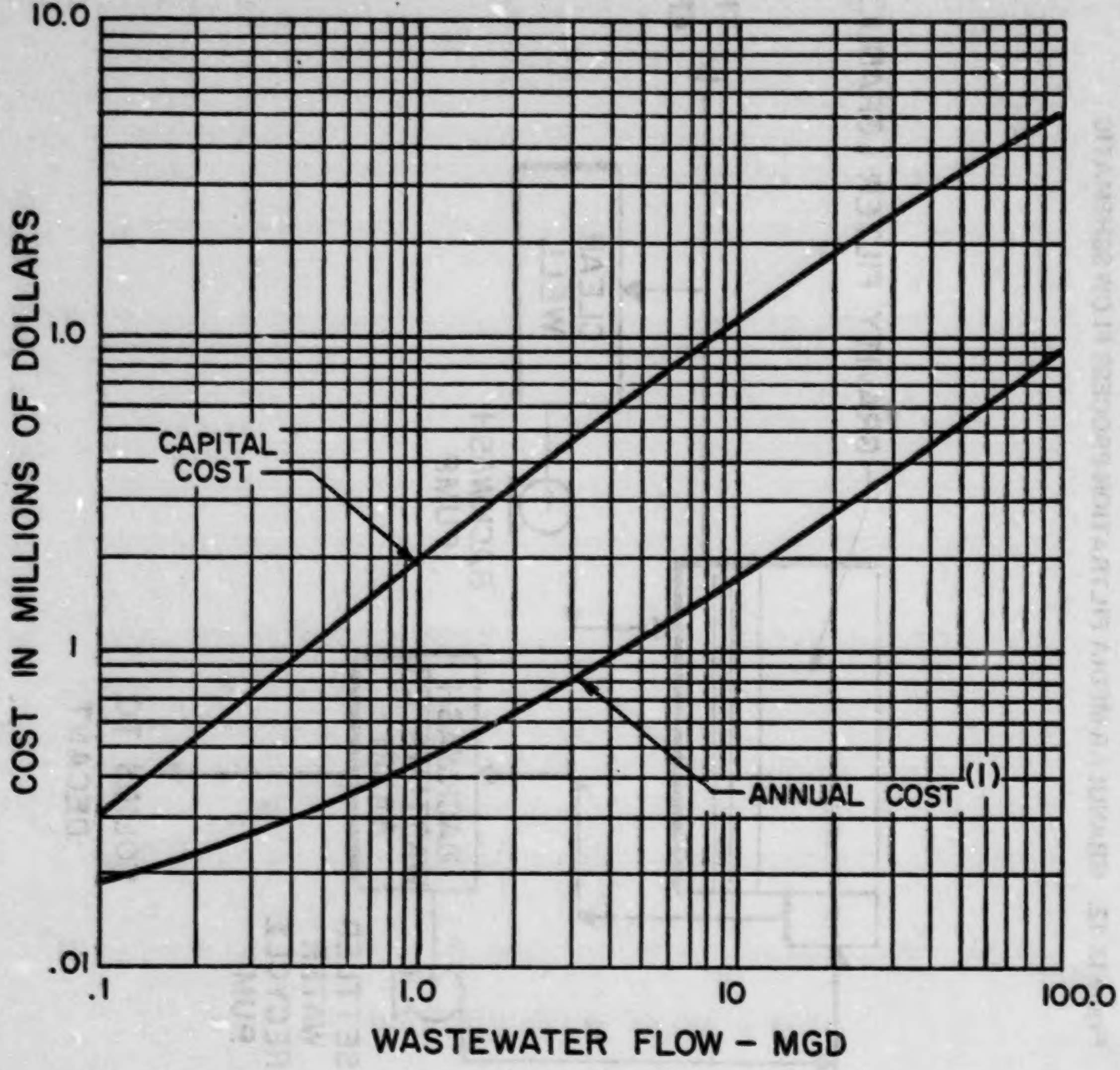

WASTEWATER INFLUENT
GRAVITY FILTER (GRANULAR MEDIA)
TREATED EFFLUENT
CLEAR WELL
BACKWASH PUMP
BACKWASH WASTEWATER BASIN
SETTLED WATER RECYCLE PUMP
SOLIDS TO DECANT

Figure IX-13. ORE MINE WASTEWATER TREATMENT GRANULAR MEDIA FILTRATION PROCESS COST CURVES



(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED

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Figure IX-14. pH ADJUSTMENT FLOW SCHEMATIC

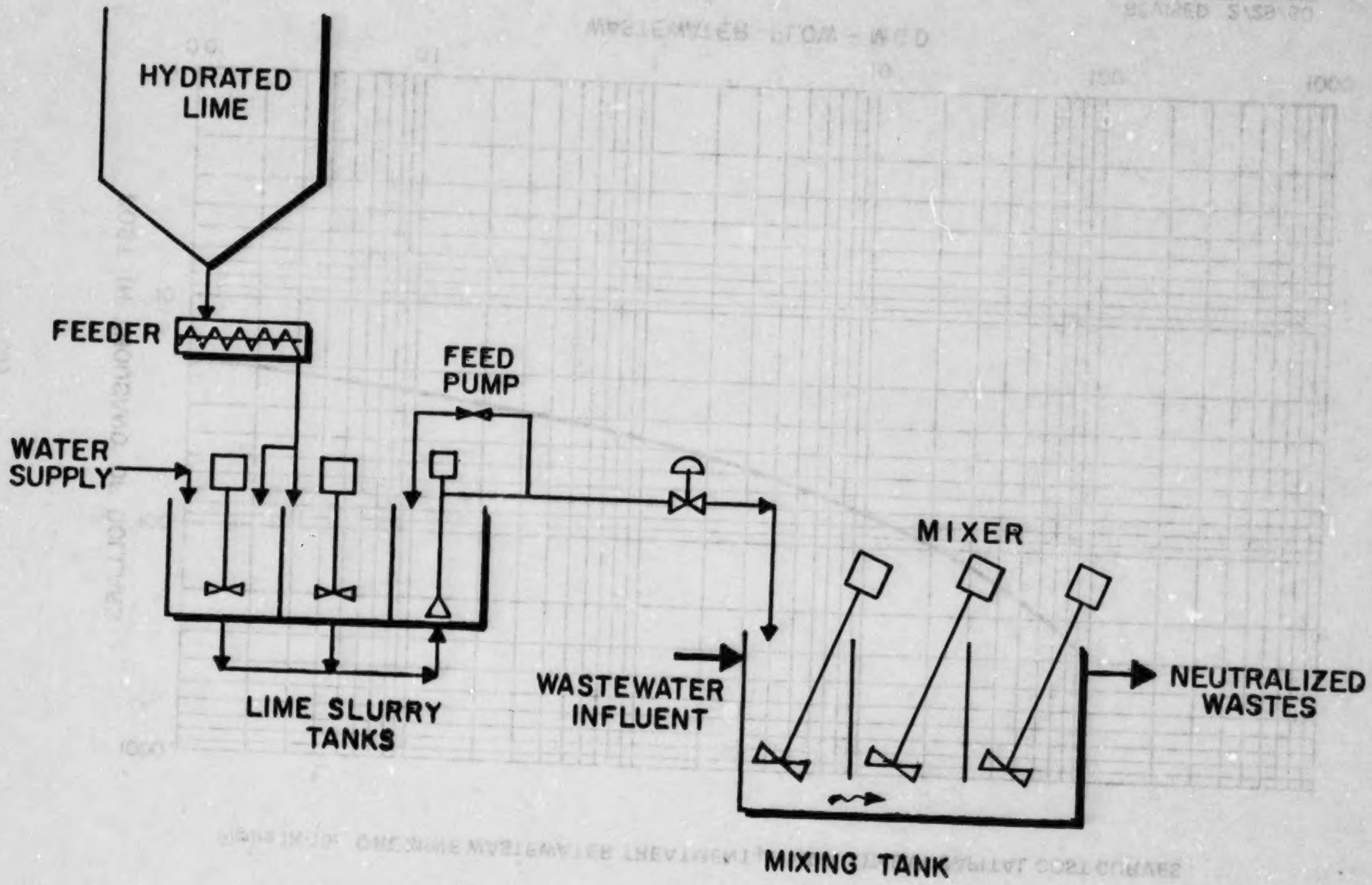
HYDRATED LIME
FEEDER
WATER SUPPLY
FEED PUMP
LIME SLURRY TANKS
MIXER
WASTEWATER INFLUENT
NEUTRALIZED WASTES
MIXING TANK

Figure IX-15. ORE MINE WASTEWATER TREATMENT pH ADJUSTMENT CAPITAL COST CURVES

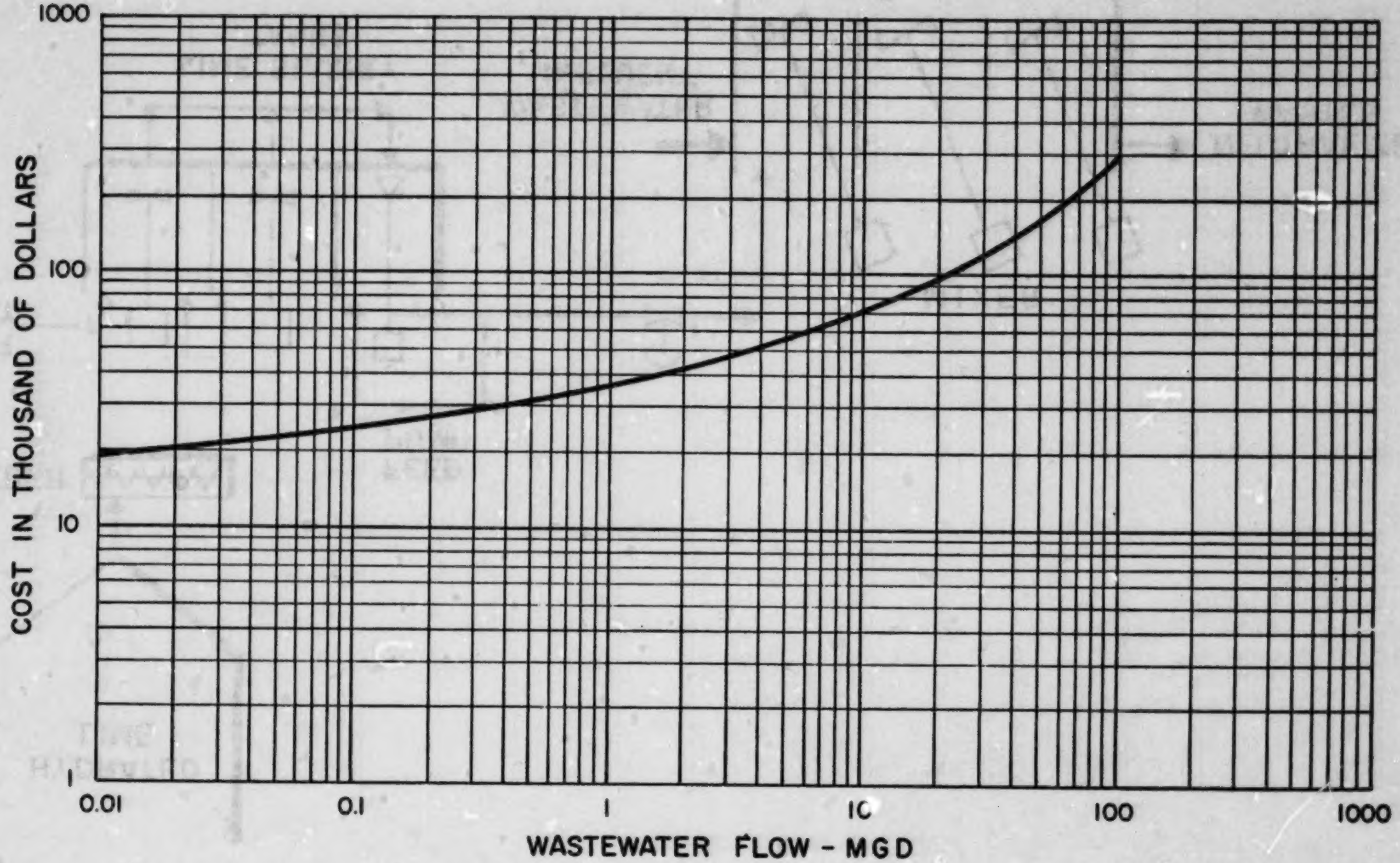

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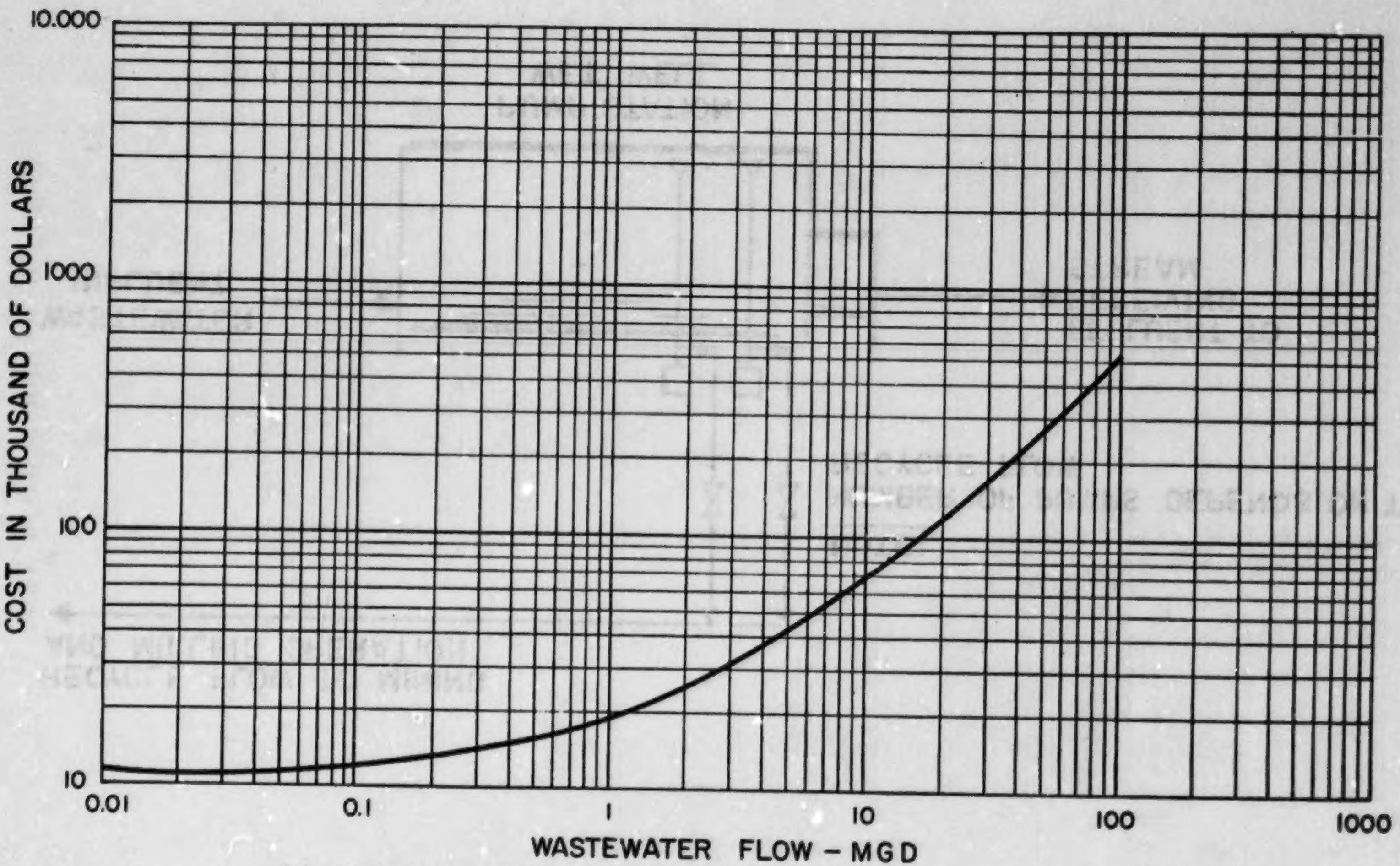
Figure IX-16. ORE MINE WASTEWATER TREATMENT pH ADJUSTMENT ANNUAL(1) COST CURVES



(1) WASTEWATER SAMPLE ANALYSIS NOT INCLUDED.

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Figure IX-17. WASTEWATER RECYCLE FLOW SCHEMATIC

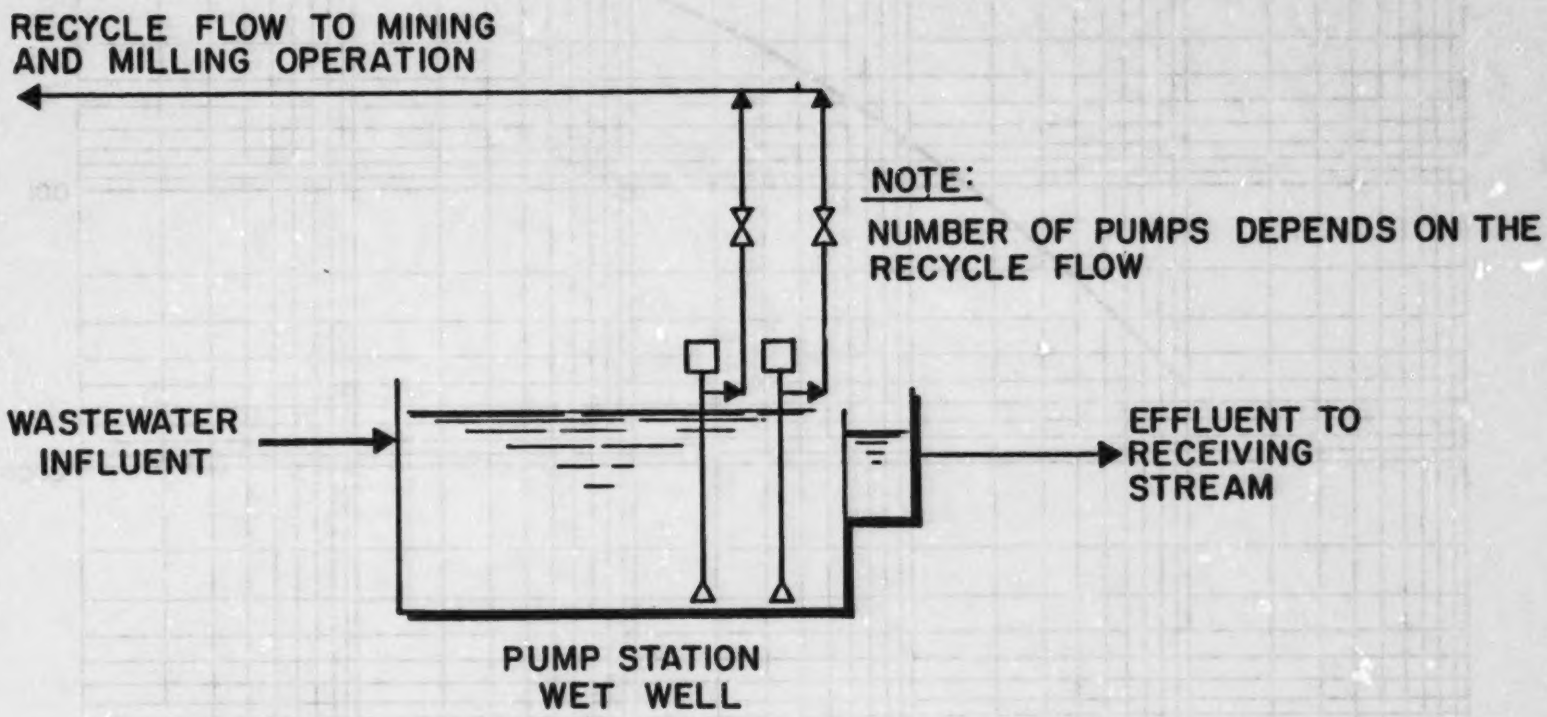

RECYCLE FLOW TO MINING AND MILLING OPERATION
NOTE:
NUMBER OF PUMPS DEPENDS ON THE RECYCLE FLOW
WASTEWATER INFLUENT
EFFLUENT TO RECEIVING STREAM
PUMP STATION WET WELL

Figure IX-18. ORE MINE WASTEWATER TREATMENT RECYCLING CAPITAL COST CURVES

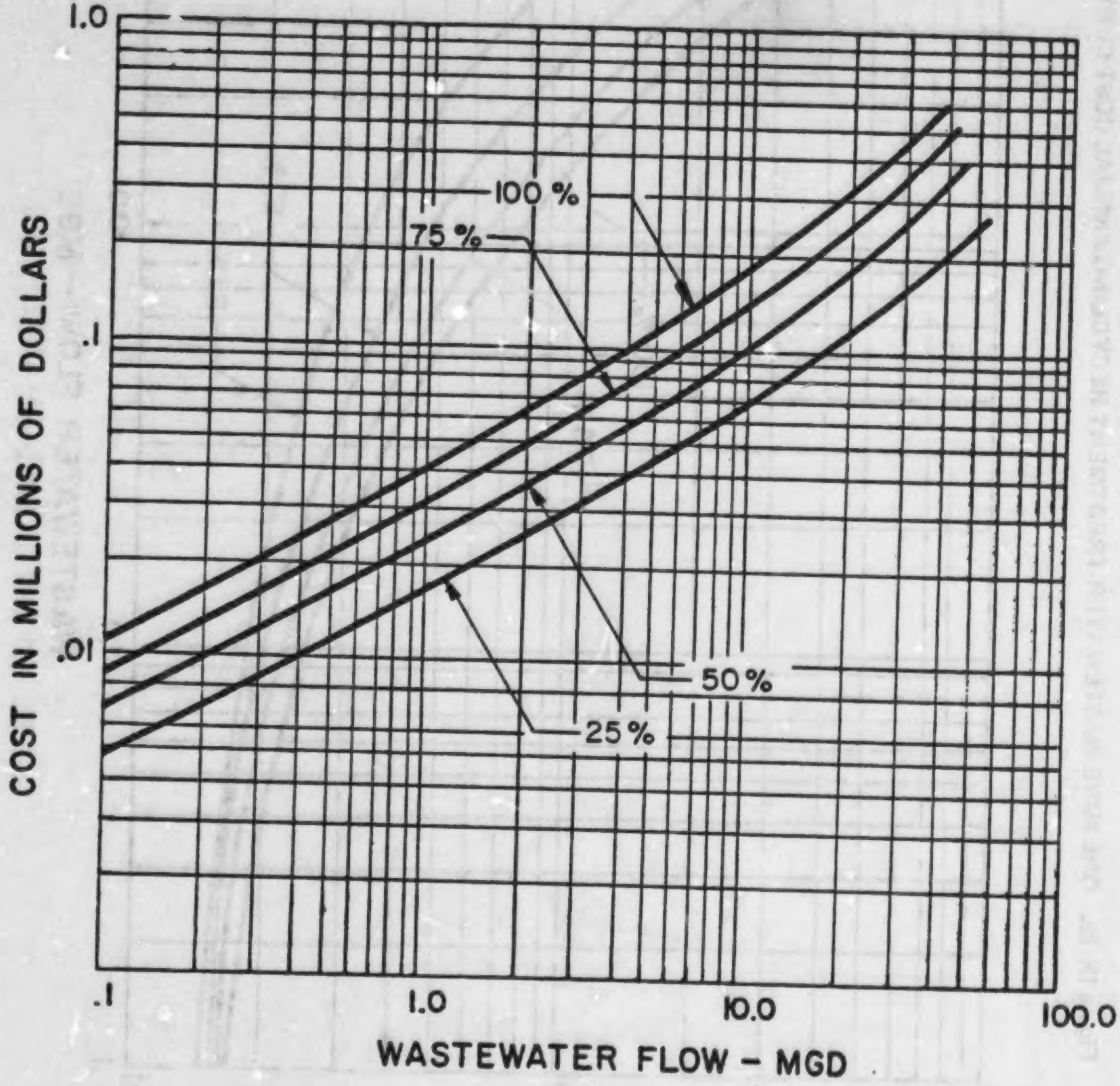
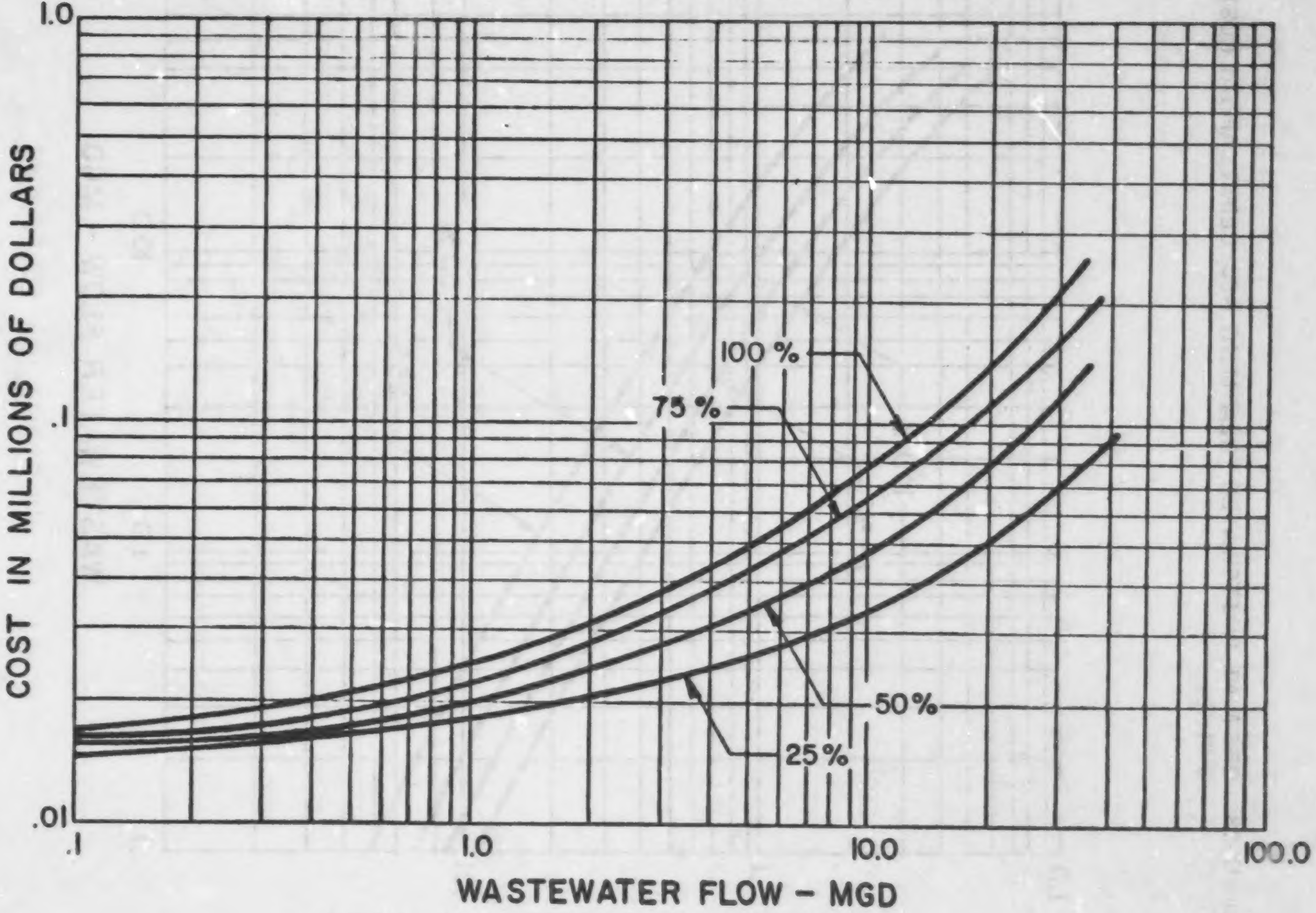


Figure IX-19. ORE MINE WASTEWATER TREATMENT RECYCLING ANNUAL COST CURVES



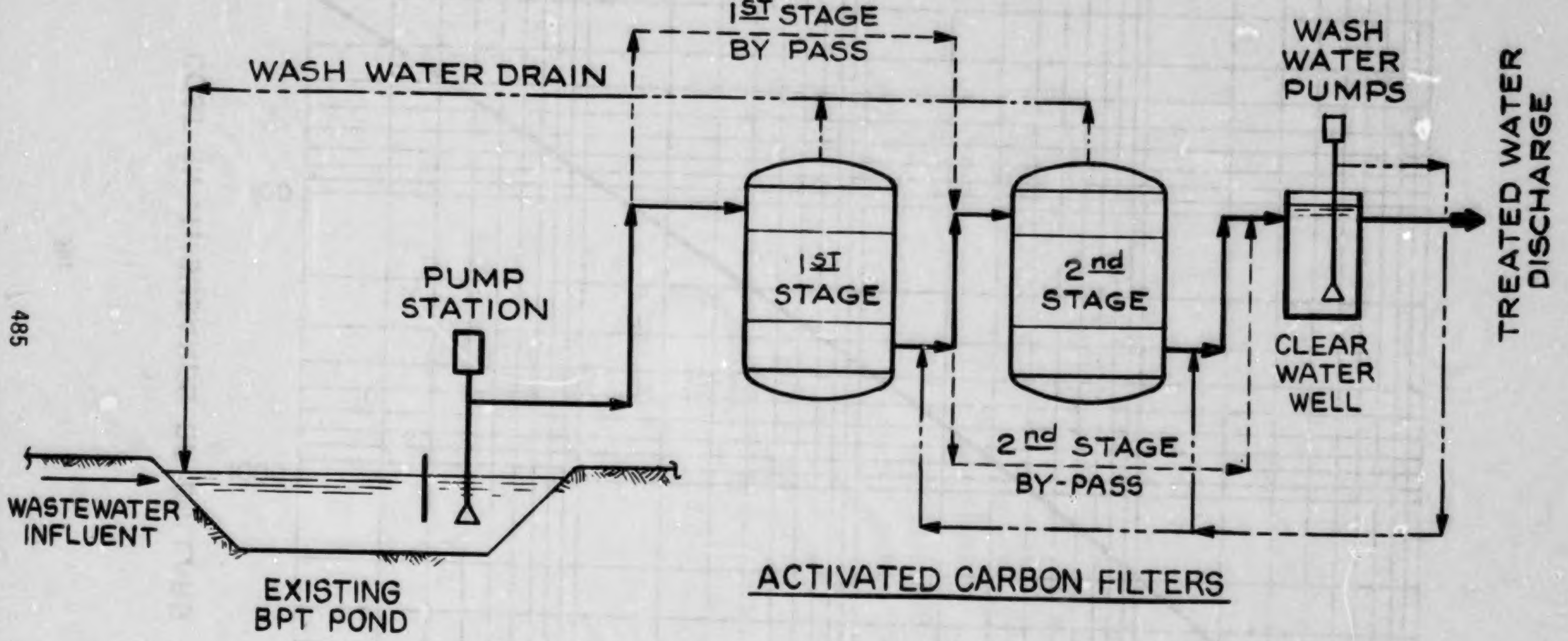


Figure IX-20. ACTIVATED CARBON ADSORPTION FLOW SCHEMATIC

1ST STAGE BY PASS
WASH WATER DRAIN
WASH WATER PUMPS
TREATED WATER DISCHARGE
PUMP STATION
1ST STAGE
2nd STAGE
CLEAR WATER WELL
2nd STAGE BY-PASS
WASTEWATER INFLUENT
EXISTING BPT POND
ACTIVATED CARBON FILTERS

Figure IX-21. ACTIVATED CARBON ADSORPTION CAPITAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE AND PRECIOUS METAL ORE MILLS

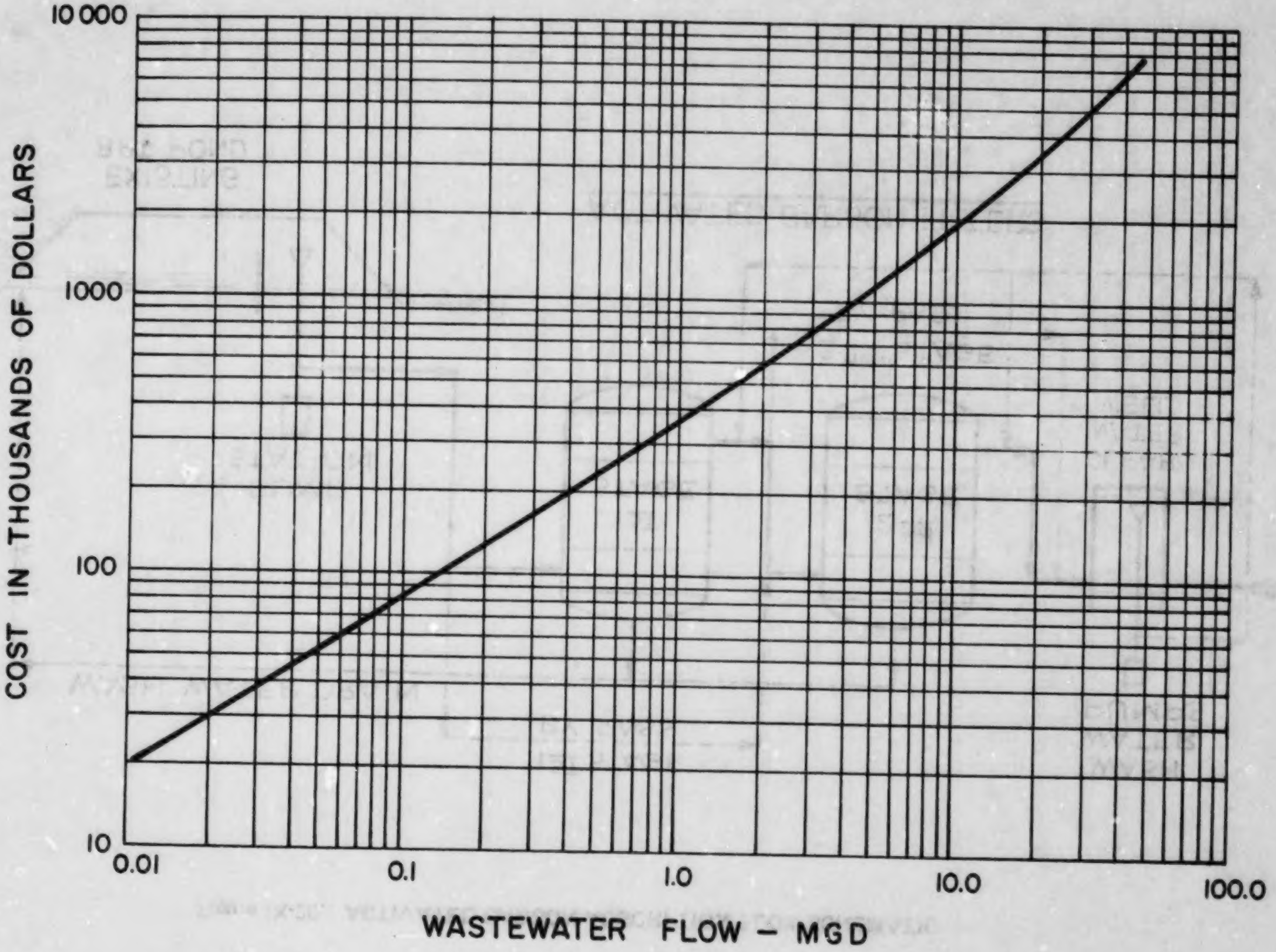
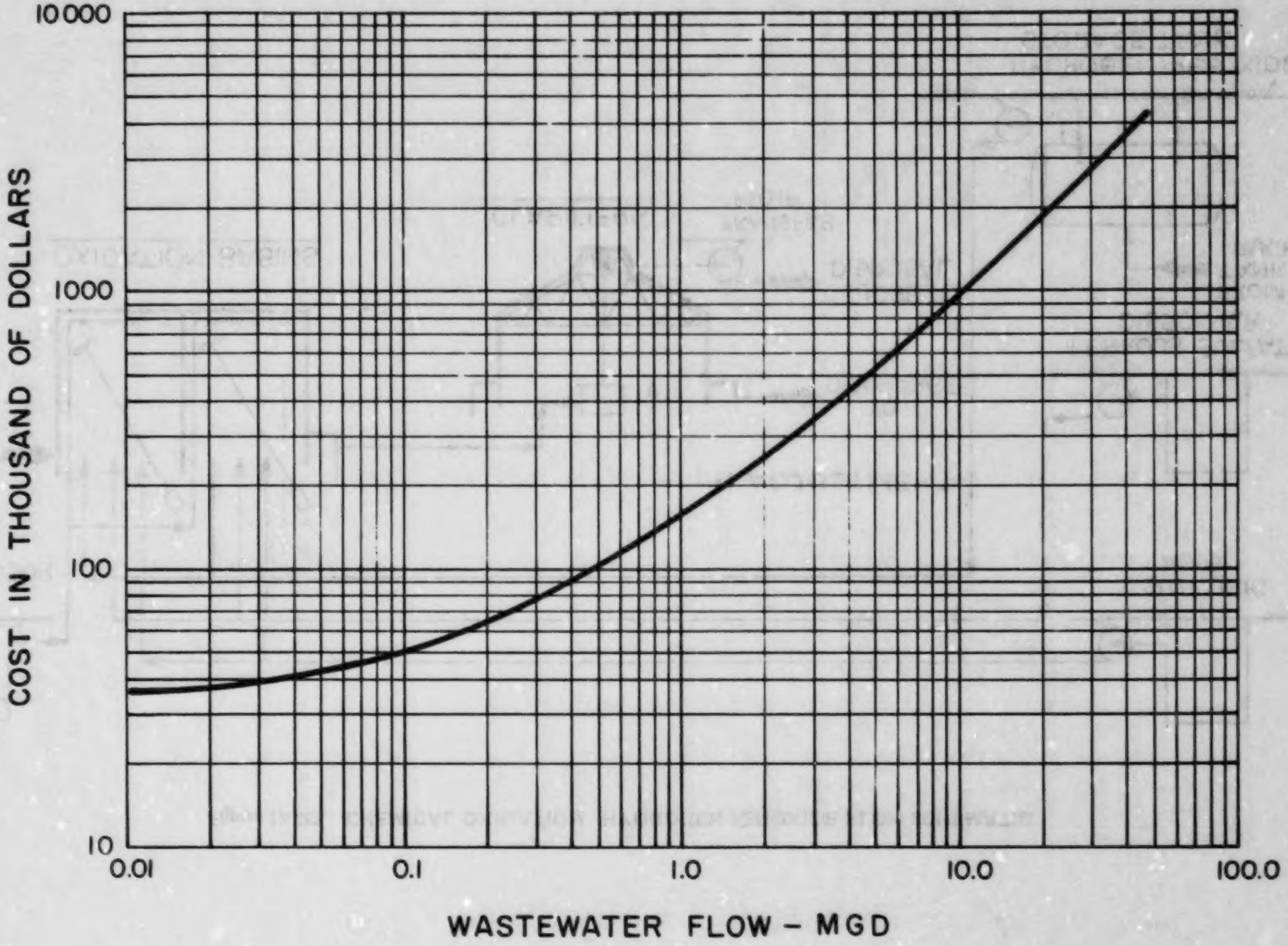


Figure IX-22. ACTIVATED CARBON ADSORPTION ANNUAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE AND PRECIOUS METAL ORE MILLS



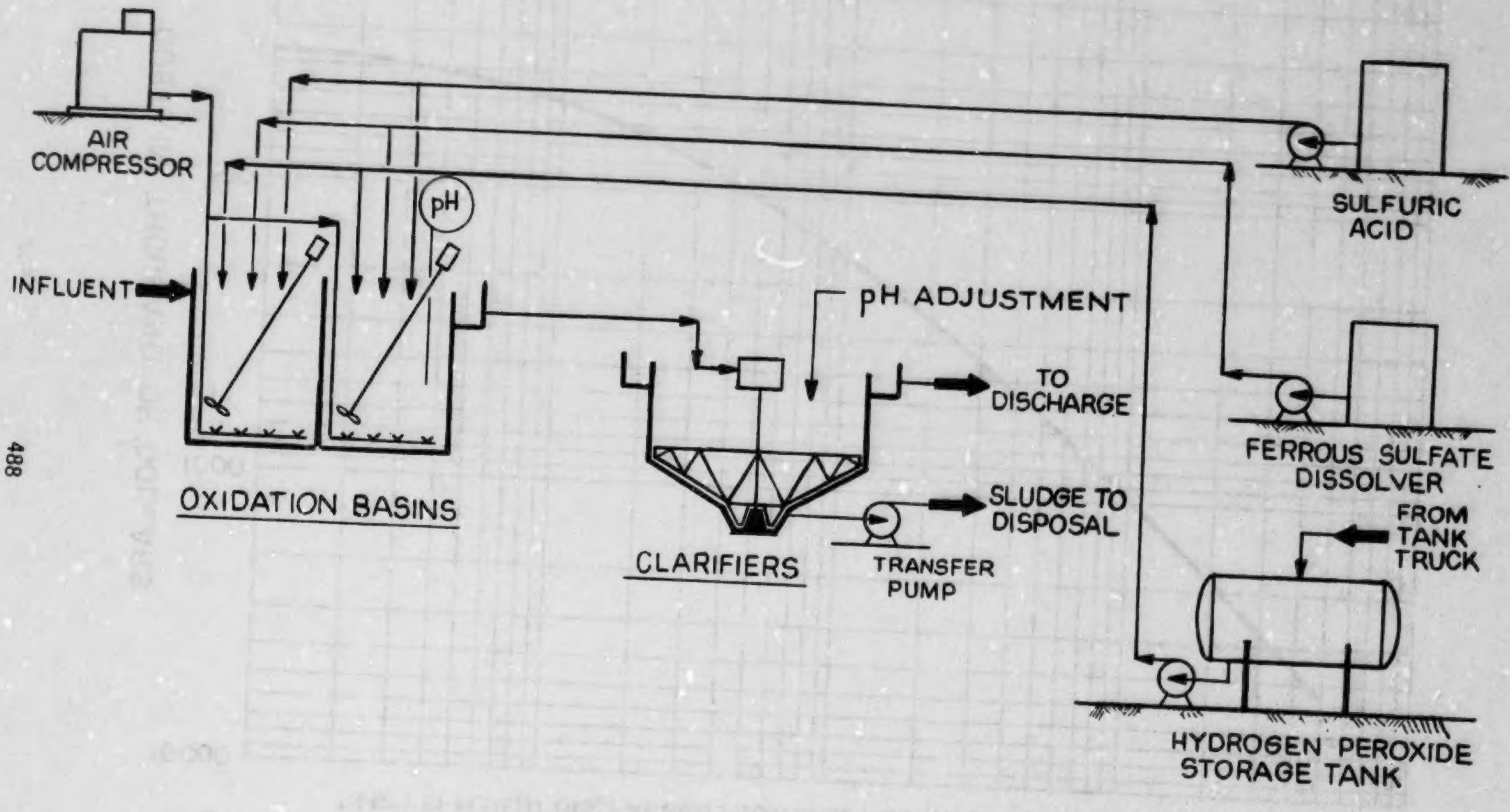

AIR COMPRESSOR
INFLUENT
pH
OXIDATION BASINS
pH ADJUSTMENT
TO DISCHARGE
SLUDGE TO DISPOSAL
CLARIFIERS
TRANSFER PUMP
SULFURIC ACID
FERROUS SULFATE DISSOLVER
FROM TANK TRUCK
HYDROGEN PEROXIDE STORAGE TANK

Figure IX-23. CHEMICAL OXIDATION - HYDROGEN PEROXIDE FLOW SCHEMATIC

Figure IX-24. HYDROGEN PEROXIDE TREATMENT CAPITAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE & PRECIOUS METAL ORE MILLS

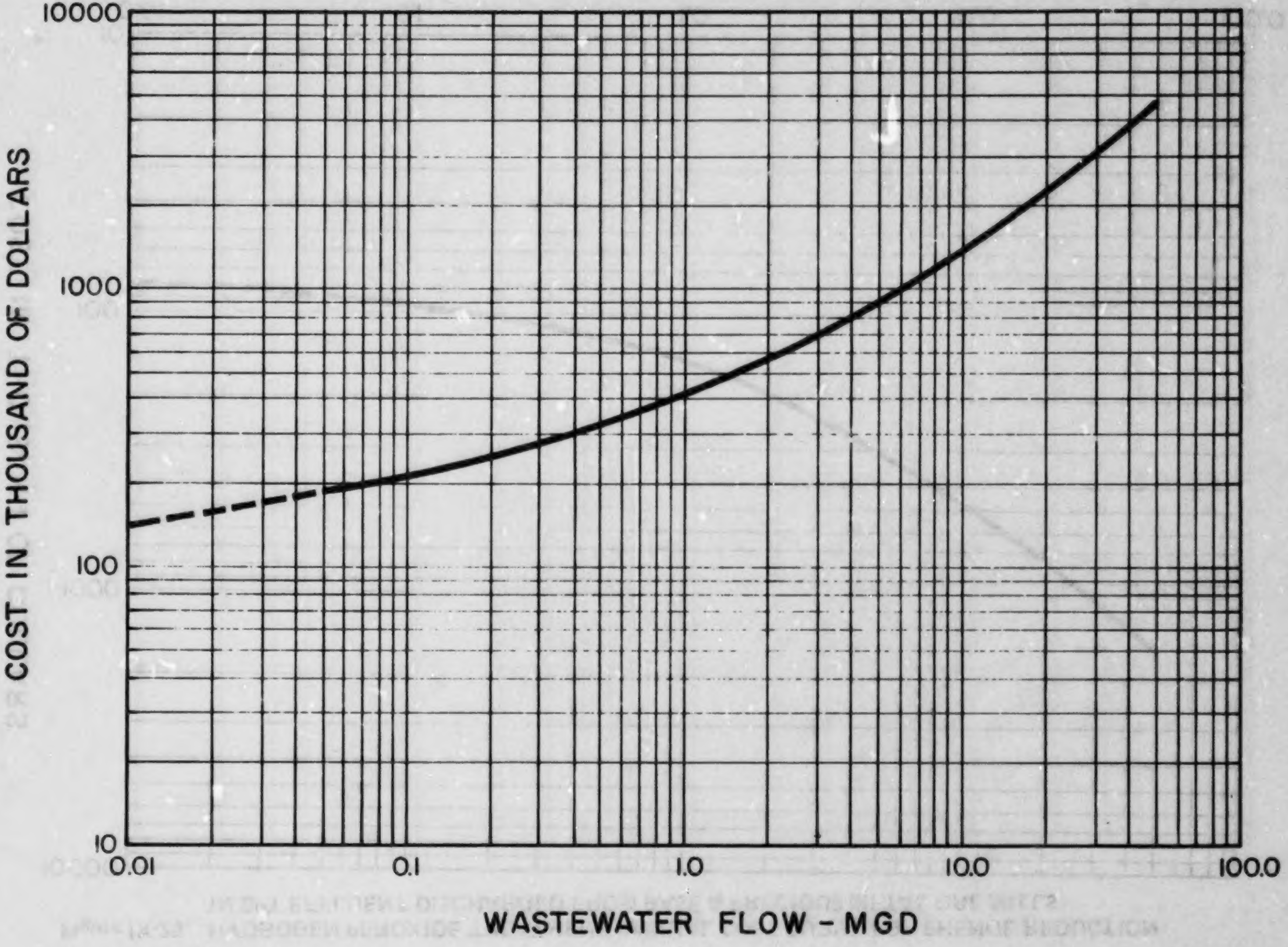


Figure IX-25. HYDROGEN PEROXIDE TREATMENT ANNUAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE & PRECIOUS METAL ORE MILLS

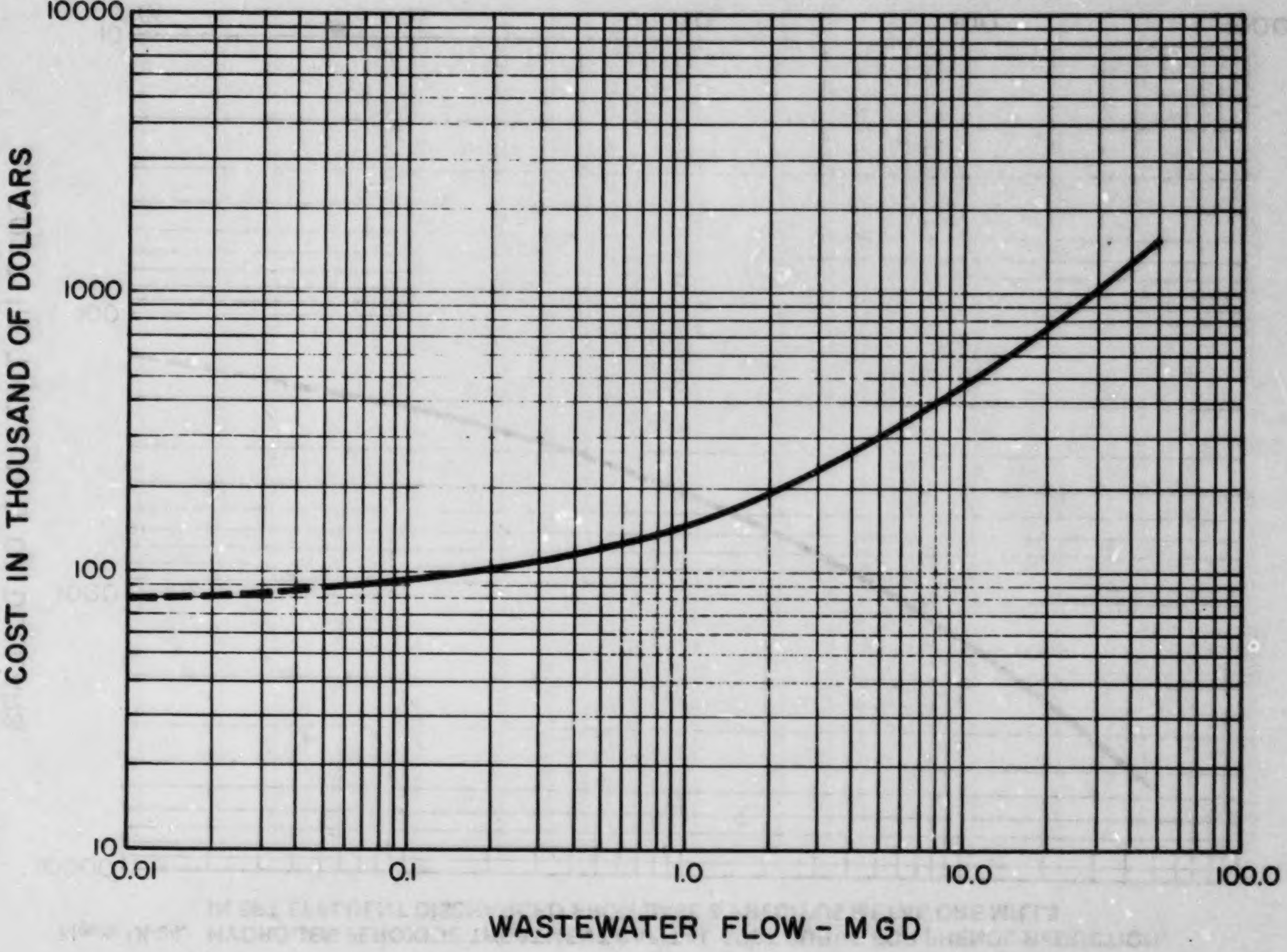


Figure IX-26. CHEMICAL OXIDATION-CHLORINE DIOXIDE FLOW SCHEMATIC

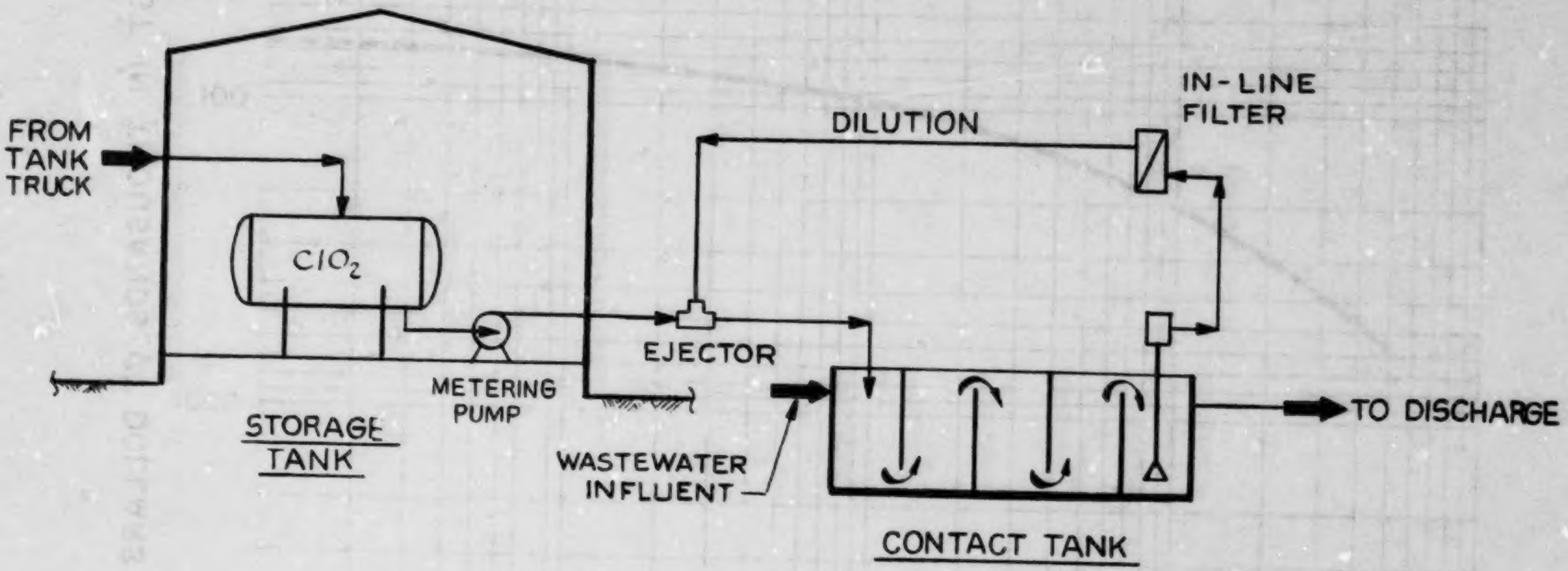

FROM
TANK
TRUCK
ClO_2
STORAGE TANK
METERING PUMP
EJECTOR
DILUTION
IN-LINE FILTER
WASTEWATER INFLUENT
CONTACT TANK
TO DISCHARGE

Figure IX-27. CHLORINE DIOXIDE CAPITAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE AND PRECIOUS METAL ORE MILLS

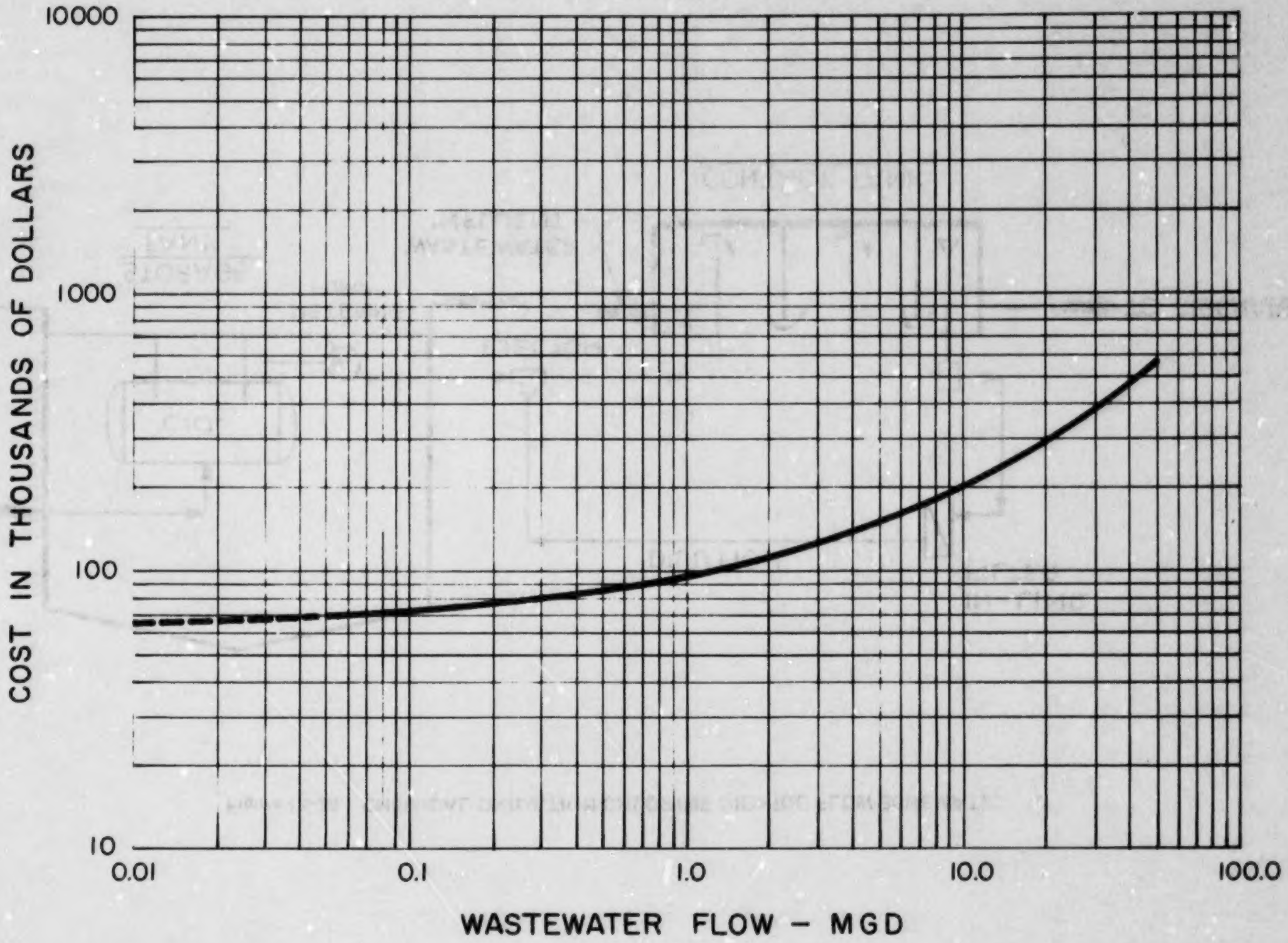


Figure IX-28. CHLORINE DIOXIDE ANNUAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE AND PRECIOUS METAL ORE MILLS

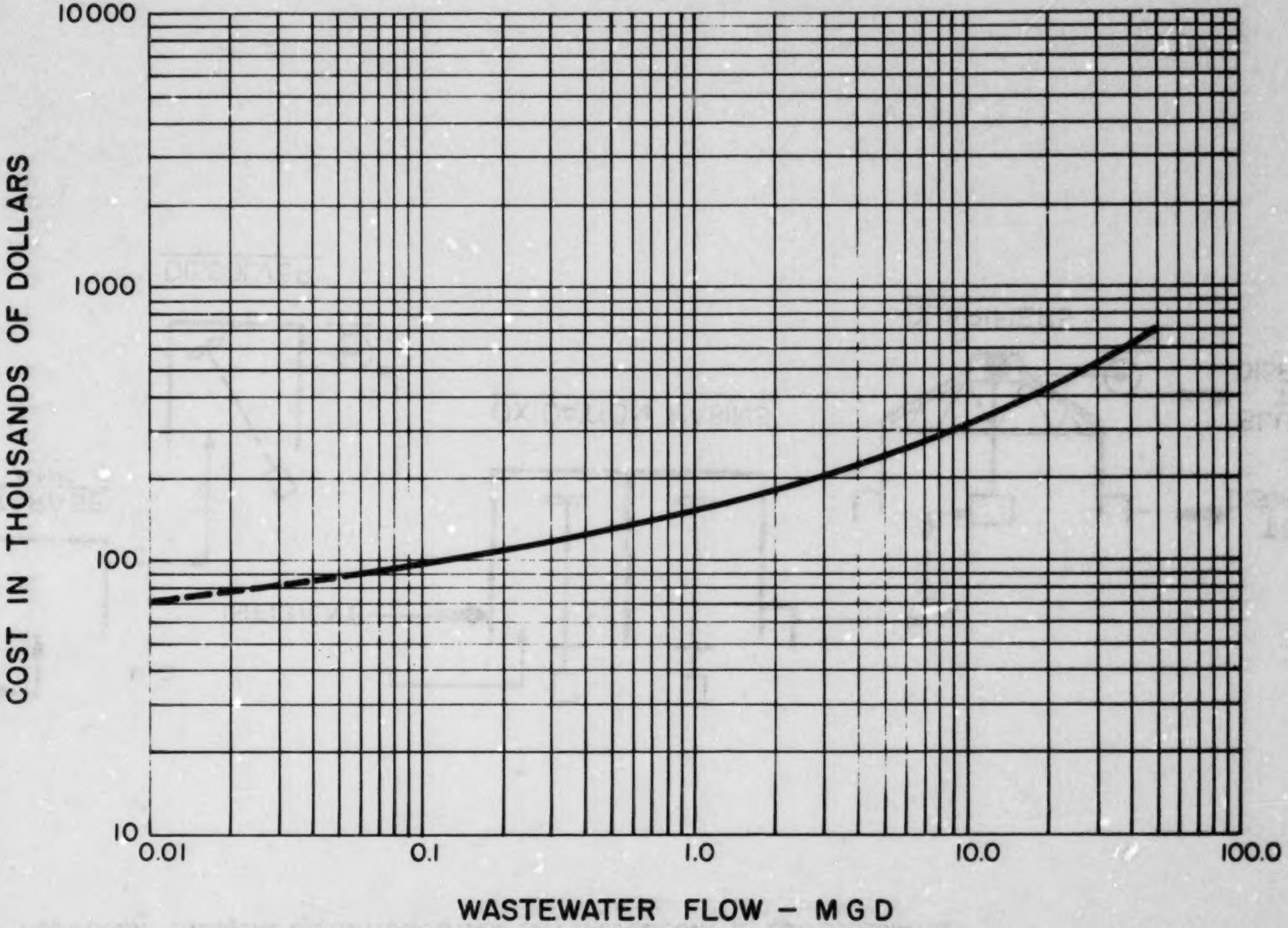


Figure IX-29. CHEMICAL OXIDATION-POTASSIUM PERMANGANATE FLOW SCHEMATIC

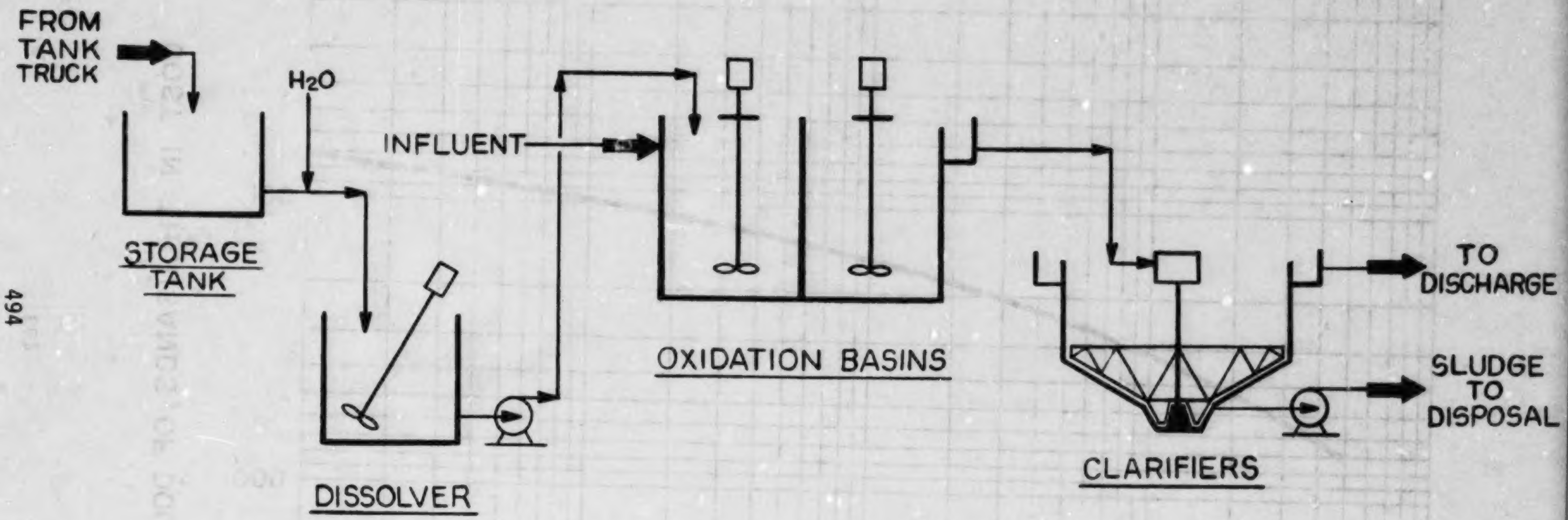

FROM
TANK
TRUCK
H2O
STORAGE TANK
INFLUENT
DISSOLVER
OXIDATION BASINS
CLARIFIERS
TO DISCHARGE
SLUDGE TO DISPOSAL

Figure IX-30. POTASSIUM PREMANGANATE TREATMENT CAPITAL COST CURVE FOR PHENOL REDUCTION IN BPT EFFLUENT DISCHARGED FROM BASE AND PRECIOUS METAL ORE MILLS

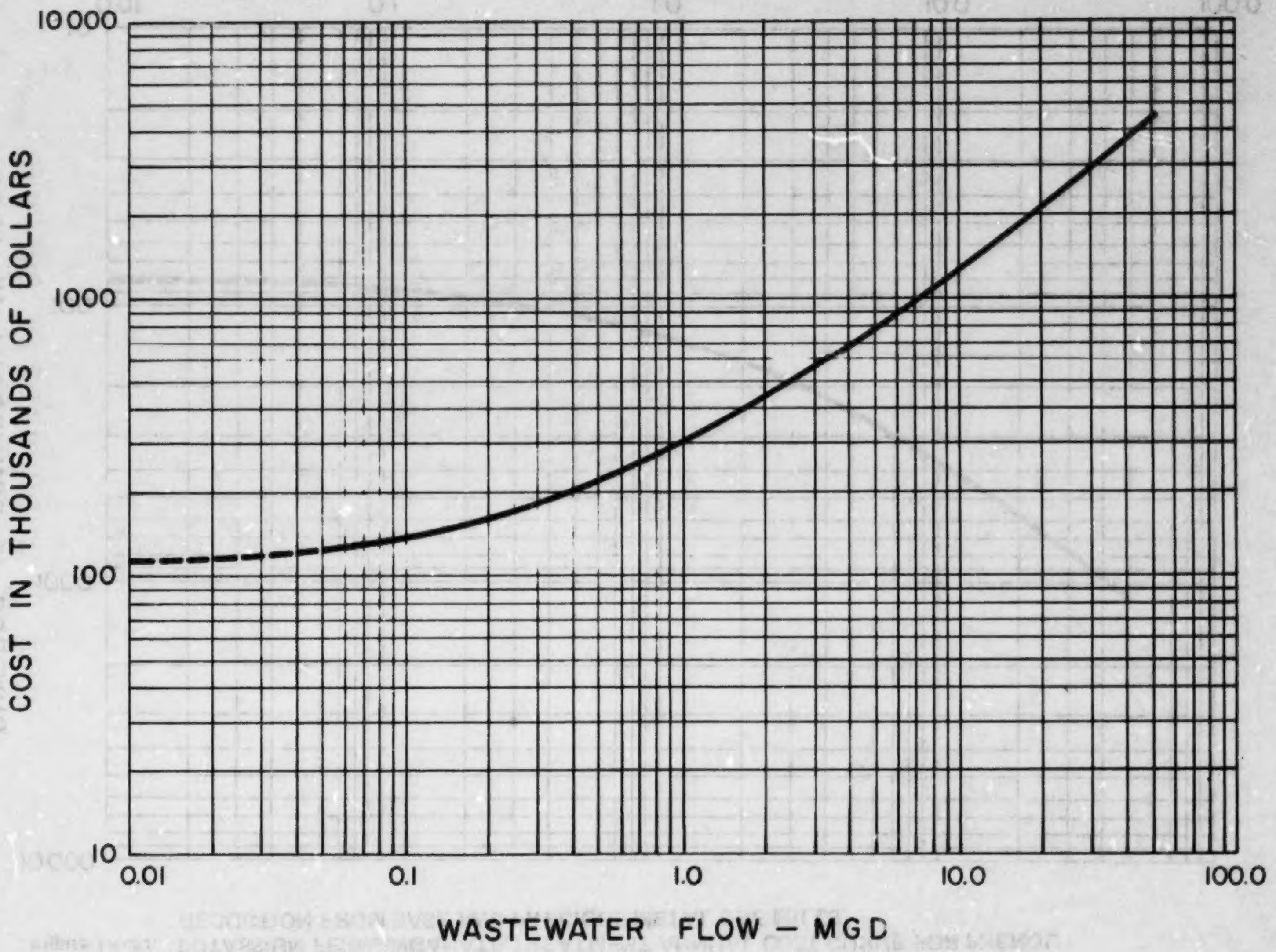
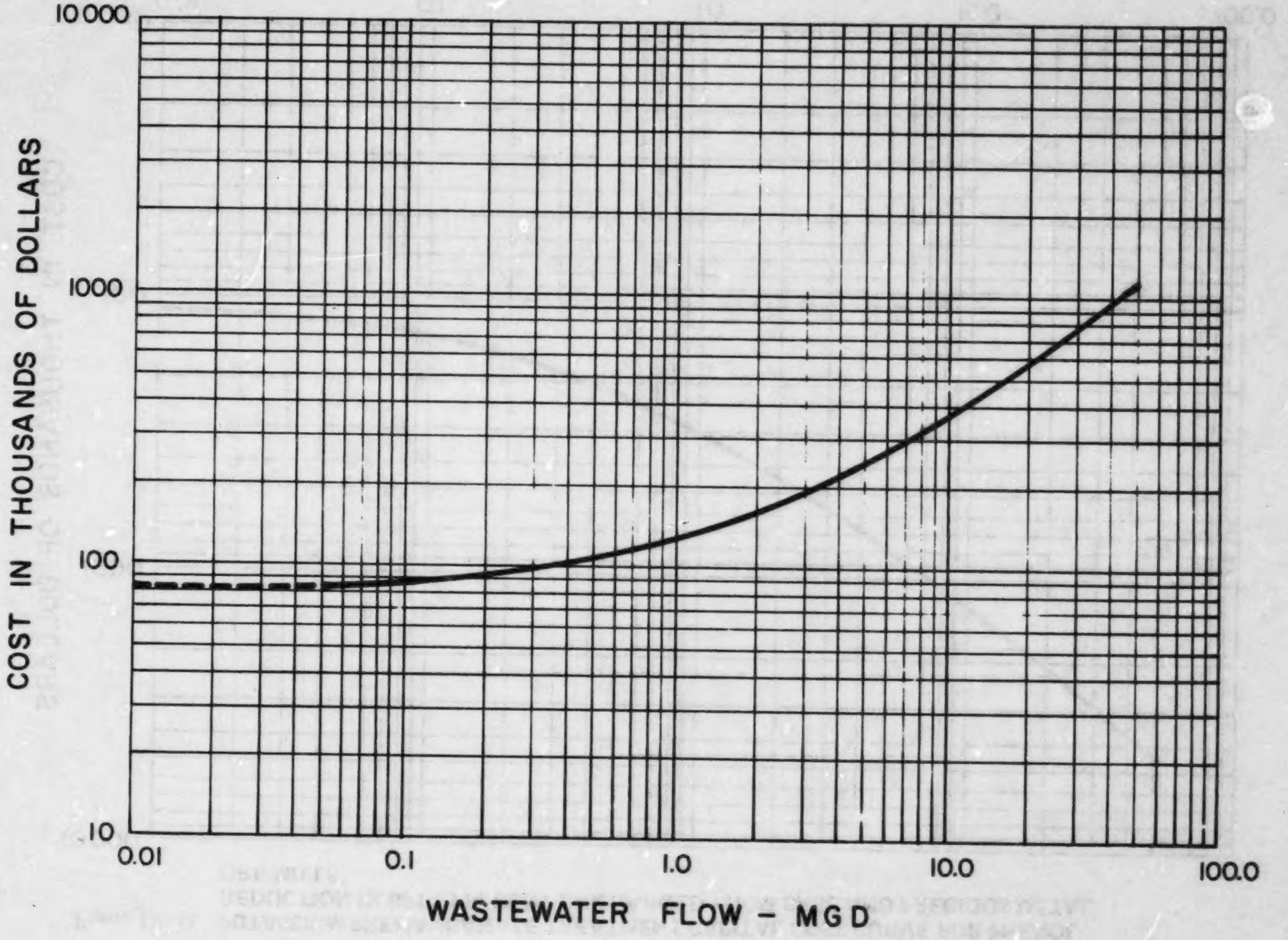


Figure IX-31. POTASSIUM PERMANGANATE TREATMENT ANNUAL COST CURVE FOR PHENOL REDUCTION FROM BASE AND PRECIOUS METAL ORE MILLS



SECTION X

BEST AVAILABLE TECHNOLOGY ECONOMICALLY ACHIEVABLE

The effluent limitations which must be achieved by 1 July 1984, are based on the best control and treatment technology employed by a specific point source within the industrial category or subcategory, or by another industry where it is readily transferable. Emphasis is placed on additional treatment techniques applied at the end of the treatment systems currently employed for BPT, as well as improvements in reagent control, process control, and treatment technology optimization.

Input to BAT selection includes all materials discussed and referenced in this document. As discussed in Section VI, nine sampling and analysis programs were conducted to evaluate the presence/absence of the pollutants (toxic, conventional, and nonconventional). A series of pilot-scale treatability studies was performed at several locations within the industry to evaluate BAT alternatives. Where industry data were available for BAT level treatment alternatives, they were also evaluated.

Consideration was also given to:

1. Age and size of facilities and wastewater treatment equipment involved

2. Process(es) employed and the nature of the ores

3. Engineering aspects of the application of various types of control and treatment techniques

4. In-process control and process changes

5. Cost of achieving the effluent reduction by application of the alternative control or treatment technologies

6. Non-water quality environmental impacts (including energy requirements)

This level of technology also considers those plant processes and control and treatment technologies which at pilot-plant and other levels have demonstrated both technological performance and economic viability at a level sufficient to justify investigation.

The Clean Water Act requires consideration of costs in BAT selection, but does not require a balancing of costs against effluent reduction benefits (see Weyerhaeuser v. Costle, 11 ERC 2149 (DC Cir. 1978)). In developing the proposed BAT, however, EPA has given substantial weight to the reasonableness of costs and reduction of discharged pollutants. The Agency has considered the volume and nature of discharges before and after application of BAT alternatives, the general environmental effects of the pollutants, and the costs and economic impacts of the required pollution control levels. The regulations proposed are, in fact, based on the application of what the Agency deems to be Best Available Control Technology Economically Achievable, with primary emphasis on significant effluent reduction capability.

The options considered are limited only by their ability to meet BPT Effluent Guidelines (as a minimum), technical feasibility in the particular subcategory, and obviously extreme (high) cost. The options presented represent a range of costs so as to assure that affordable alternatives remain after the economic analysis.

SUMMARY OF BEST AVAILABLE TECHNOLOGY

Zero discharge limitations are established for the following subcategories and subparts:

- Iron Ore
  - Mills in the Mesabi Range
- Mercury Ore
  - Mills
- Cu, Pb, Zn, Au, Ag, Pt, and Mo Ores
  - Mills using the cyanidation process or the amalgamation process to recovery gold or silver
  - Mills and mine areas that use leaching processes to recover copper

Subcategories and subparts permitted to discharge subject to limitations are:

Subcategory and Subpart	Nonconventional Pollutants Controlled	Toxics Controlled
Iron Ore		
Mine Drainage	Fe (dissolved)	
Mills (physical methods)	Fe (dissolved)	

Subcategory and Subpart	Nonconventional Pollutants Controlled	Toxics Controlled
Aluminum Ore		
Mine Drainage	Fe, Al	
Uranium, Radium, and Vanadium Ores		
Mine Drainage	COD, Ra226 (dissolved) Ra226 (total), U	Zn
Mercury Ore		
Mine Drainage		Hg
Titanium Ore		
Mine Drainage	Fe	
Mills		Zn
Dredges	Fe	
Tungsten Ore		
Mine Drainage		Cd, Cu, Zn
Mills		Cd, Cu, Zn
Cu, Pb, Zn, Au, Ag, Pt, and Mo Ores		
Mine Drainage (not placer mining)		Cd, Cu, Zn, Pb, Hg,
Mills (froth flotation)		Cd, Cu, Zn, Pb, Hg,

Subcategory and Subpart Controlled Controlled

The specific effluent limitations guidelines for the subcategories and subparts permitted to discharge are:

Toxic Pollutants	Daily Maximum mg/l	30-Day Average mg/l
Copper	0.30	0.15
Zinc	1.0 (1.5)*	0.5 (0.75)*
Lead	0.6	0.3
Mercury	0.002	0.001
Cadmium	0.10	0.05

Nonconventional Pollutants	Daily Maximum mg/l	30-Day Average mg/l
Iron (dissolved)	2.0	1.0
Iron (total)	2.0	1.0
Aluminum	2.0	1.0
COD	--	500
Radium 226 (dissolved)	10 (pCi/l)	3 (pCi/l)
Radium 226 (total)	30 (pCi/l)	10 (pCi/l)
Uranium	4	2

*Limitations applicable to mine drainage for the copper, lead, zinc, gold, silver, platinum, and molybdenum ores subcategory.

GENERAL PROVISIONS

Several items of discussion apply to options in more than one subcategory. To avoid repetition, these items are discussed here and referred to in the discussion of the options.

Relief From No Discharge Requirement

Facilities which are not allowed to discharge under "normal" conditions may do so as a result of:

1. An overflow or increase in volume from a precipitation event if the facility is designed, constructed and maintained to contain a 10-year, 24-hour rainfall design (storm provision); or

2. If they are located in a "net precipitation" area.

These exemptions are discussed in greater detail below.

Storm Provision

The Agency recognizes that relief is necessary as a practical matter for many discharges within the ore mining and dressing point source category during and immediately after some precipitation events. It would be unreasonable to require facilities to construct retention structures and treatment facilities to handle runoff resulting from extreme rainfall conditions which could statistically occur only rarely. Further, it must be emphasized that the regulations for the ore mining and dressing point source category do not require any specific treatment technique, construction activity, or other process for the reduction of pollution. The effluent limitations guidelines limit the concentration of pollutants which may be discharged, while allowing for an excursion from the normal requirements when precipitation causes an overflow or increase in the volume of a discharge from a facility properly designed, constructed, and maintained to contain or treat a 10-year, 24-hour rainfall.

This relief applies to the excess volume caused by precipitation or snow melt, and the resulting increase in flow or shock flow to the settling facility or treatment facility. While there has been criticism of the relief adopted by the Agency, the few alternatives suggested by environmental groups and industry are substantially less satisfactory in light of the data available to the Agency. This is discussed in detail in the preamble to the BPT regulation (43 FR 29771) and in the clarification of regulations (44 FR 7953).

The general relief in the BPT regulation states that:

> "Any excess water, resulting from rainfall or snow melt, discharged from facilities designed, constructed and maintained to contain or treat the volume of water which would result from a 10-year, 24-hour pecipitation event shall not be subject to the limitations set forth in 40 CFR 440." 43 FR at 29777-78, 440.81(c)(1978).
>
> The term "ten-year, 24-hour precipitation event" is defined, in turn, as:
>
> "the maximum 24-hour precipitation event with a probable reoccurrence interval of once in 10 years as defined by the National Weather Service and Technical Paper No. 40, 'Rainfall Frequency Atlas of the U.S.,' May 1961, and subsequent amendments, or equivalent regional or rainfall probability information developed therefrom." 43 FR at 29778, 440.82(d).

Under BAT, similar relief is granted: for existing sources that are designed, constructed, and maintained to contain or treat the maximum volume of process wastewater discharged in a 24-hour period, including the volume which would result from a 10-year, 24-hour precipitation event, or snow melt of equal volume, any excess wastewater discharged shall not be subject to the limitations set forth in 40 CFR 440.

In determining the maximum volume of wastewater which would result from a 10-year, 24-hour precipitation event at any facility, the volume must include the volume that would result from runoff from all areas contributing runoff to the individual treatment facility, i.e., all runoff that is not diverted from the active mining area, runoff which is not diverted from the mill area, and other runoff that is allowed to commingle with the influent to the treatment system.

In general, the following will apply in granting relief:

1. The exemption as stated in the rule is available only if it is included in the operator's permit. Many existing permits have exemptions or relief clauses stating requirements other than those set forth in the rule. Such relief clauses remain binding unless and until an operator requests a modification of his permit to include the exemption as stated in the rule.

2. The storm provision is an affirmative defense to an enforcement action. Therefore, there is no need for the permitting authority to evaluate each tailings pond or treatment facility now under permit.

3. Relief can be granted to deep mine, surface mine, and ore mill discharges.

4. Relief is granted as an exemption to the requirements for normal operating conditions (i.e., without overflow, increase in volume of discharge, or discharge from a by-pass system caused by precipitation) with respect to an increase in flow caused by surface runoff only.

5. Relief can be granted for discharges during and immediately after any precipitation or snow melt. The intensity of the event is not specified.

6. The relief does not grant, nor is it intended to imply the option of ceasing or reducing efforts to contain or treat the runoff resulting from a precipitation event or snow melt. For example, an operator does not have the option of turning off the lime feed to a facility at the start of or during a precipitation event, regardless of the design and construction of the wastewater facility. The operator must continue to operate his facility to the best of his ability.

7. Under the regulation, relief is granted from all effluent limitations contained in BAT.

8. In general, relief can not be granted to treatment facilities which employ clarifiers, thickeners, or other mechanically aided settling devices. The use of mechanically aided settling usually is restricted to discharges which are not affected by runoff.

9. In general, the relief was intended for discharges from tailings ponds, settling ponds, holding basins, lagoons, etc. that are associated with and part of treatment facilities. The relief will most often be based on the construction and maintenance of these settling facilities to "contain" a volume of water.

10. The term "treat" for facilities allowed to discharge means the wastewater facility was designed, constructed, and maintained to meet the daily maximum effluent limitations for the maximum flow that would result from a 10-year, 24-hour rainfall. The operator has the option to "treat" the volume of water that would result from a 10-year, 24-hour rainfall in order to qualify for the rainfall exemption, or as mentioned in paragraph 9 above, the second option is to "contain" the volume of wastewater.

11. The term "maintain" is intended to be synonymous with "operate." The facility must be operated at the time of the precipitation event to contain or treat the specified volume of wastewater. Specifically, in making a determination of the ability of a facility to contain a volume of wastewater, sediment and sludge must not be permitted to accumulate to such an extent that the facility cannot in fact hold the volume of wastewater resulting from a 10-year, 24-hour rainfall. That is, sediment and sludge must be removed as required to maintain the specific volume of wastewater required by the rule, or the embankment must be built up or graded to maintain a specific volume of wastewater required by the rule.

12. "Contain" and "maintain" for facilities which are allowed to discharge do not mean providing for draw down of the pool level of the facility allowed to discharge. As an example, the volume can be determined from the top of the stage of the highest dewatering device to the bottom of the pond at the time of the precipitation event. There is no requirement that relief be based on the facility which is allowed to discharge being emptied of wastewater prior to the rainfall or snow melt upon which the relief is granted. The term "contain" for facilities which are allowed to discharge means the wastewater facility's tailings pond or settling pond was designed to include the volume of water that would result from a 10-year, 24-hour rainfall.

13. The term "contain" for facilities which are not allowed to discharge means the wastewater facility was designed,

constructed, and maintained to hold, without a point source discharge, the volume of water that would result from a 10-year, 24-hour rainfall, in addition to the normal amount of water which would be in the wastewater facility.

Net Precipitation Areas

The general relief or exemption for the requirement of "no discharge of process wastewater" as promulgated for ore mining and dressing in BPT Guidelines (43 FR 29771) states that:

> "In the event that the annual precipitation falling on the treatment facility and the drainage area contributing surface runoff to the treatment facility exceeds the annual evaporation, a volume of water equivalent to the difference between annual precipitation falling on the treatment facility and the drainage area contributing surface runoff to the treatment facility and annual evaporation may be discharged subject to the limitations set forth in paragraph (a) of the section." Paragraph (a) refers to limitations established for mine drainage.
>
> Relief for net precipitation areas is included in the BAT regulation.

Commingling Provision

The general provision as promulgated for ore mining and dressing in the BPT Guidelines (43 FR 29771) states that:

> "In the event that waste streams from various subparts or segments of subparts in part 440 are combined for treatment and discharge, the quantity or quality of each pollutant or pollutant property in the combined discharge that is subject to effluent limitations shall not exceed the quantity or quality of each pollutant or pollutant property that would have been discharged had each waste stream been treated separately. The discharge flow from a combined discharge shall not exceed the volume that would have been discharged had each waste stream been treated separately."

This consideration is also appropriate for BAT and the regulation states:

> For existing sources which as of the date of this proposal have combined for treatment waste streams from various subparts or segments of subparts in Part 440, the quantity and quality of each pollutant or pollutant property in the combined discharge that is subject to effluent limitations shall not exceed the quantity and quality of each pollutant or pollutant property that would have been discharged had each waste stream been treated separately. The discharge flow from a combined discharge shall not exceed the volume

that would have been discharged had each waste stream been treated separately.

BAT OPTIONS CONSIDERED FOR TOXICS REDUCTION

As discussed in Section VII, many toxic pollutants found in this category are related to TSS (that is, as TSS concentrations are reduced during treatment, observed concentrations of certain toxic metals are also reduced). In order to remove these toxics, suspended solid removal technologies can be used. The technologies are secondary settling, coagulation and flocculation, and granular media filtration. They are applicable throughout the category for suspended solids reduction and associated toxic metals reduction, and are discussed here to avoid repetition during description of options. Dissolved metals are not controlled further by physical treatment methods or additional suspended solids removal.

Secondary Settling

This option involves the addition of a second settling pond in series with the existing pond (as described in Section VIII). The technique is used in many ore subcategories. The most prevalent configuration is a second pond located in series with a tailings pond.

Examples of the use of secondary and tertiary settling ponds can be seen at lead/zinc Mills 3101, 3102, 3103 and at Mill 4102 (Pb/Zn/Au/Ag). This last facility uses a secondary pond to achieve an effluent level of 4 mg/l TSS, as determined during sampling (Reference 1). Secondary settling ponds (sometimes called polishing ponds) are also used in settling solids produced in the coprecipitation of radium with barium salts at uranium mines and mills. (See Section VIII, End-of-Pipe Techniques, Secondary Settling; and Historical Data Summary, lead/zinc Mills 3101, 3102, 3103, and 4102.)

Coagulation and Flocculation

Chemically aided coagulation followed by flocculation and settling is described in Section VIII. It is used by facilities in several subcategories of the industry for solids and metals reduction.

At Mine/Mill 1108, the tailing pond effluent is treated with alum followed by polymer addition and secondary settling to reduce TSS from 200 mg/l to an average of 6 mg/l. At Mine 3121, polymer addition has greatly improved the treatment system capabilities. A TSS mean concentration of 39 mg/l (range 15 to 80) has been reduced to a mean of 14 mg/l (range 4 to 34), a reduction of 64 percent. Similarly, polymer use at Mine 3130 reduced treated effluent total suspended solids concentrations from a mean of 19 mg/l (range 4 to 67 mg/l) to a mean of 2 mg/l (range of 1 to 6.2

mg/l). It should be pointed out that these effluent levels are attained by the combination of settling aids and a secondary settling pond (Section VIII).

Granular Media Filtration

This option uses granular media such as sand and anthracite to filter out suspended solids, including the associated metals (as discussed in Section VIII). This technology is used at one facility (Mine 6102) and has been pilot tested at other mines and mills and is used in other industry categories.

Granular-media filtration can consistently remove 75 to 93 percent of the suspended solids from lime-treated secondary sanitary effluents containing from 2 to 139 mg/l of suspended solids (Reference 2). In 1978, lead/zinc Mine/Mill/Smelter/Refinery 3107 was operating a pilot-scale filtration unit to evaluate its effectiveness in removing suspended solids and nonsettleable colloidal metal-hydroxide flocs from its wastewater treatment plant. Granulated slag was used as the medium in some of the tests. Preliminary data indicate that the single medium pressure filter was capable of removing 50 to 95 percent of the suspended solids and 14 to 82 percent of the metals (copper, lead and zinc) contained in the waste stream. Final suspended solids concentrations which have been obtained are within the range of 1 to 15 mg/l. Optimum filter performance was attained at lower hydraulic loadings (2.7

A full-scale multi-media filtration unit is currently in operation at molybdenum Mine/Mill 6102. The filtration system is used as treatment following settling (tailing pond), ion exchange (for molybdenum removal), lime precipitation, electrocoagulation, and alkaline chlorination. Since startup in 1978, the filtration unit has been operating at a flow of 63 liters/second (1000 gpm) and monitoring data show TSS reductions to an average of less than 5 mg/l. Zinc removal and iron reduction have also been achieved (see Treatment Technology - Section VIII).

A pilot-scale study of mine drainage treatment in Canada has also demonstrated the effectiveness of filtration (Section VIII). Polishing of clarifier overflow by sand filtration resulted in reduction of the concentration of lead and zinc (approximately 50 percent) and removal of iron (approximately 40 percent) and copper.

In addition to the above, a full-scale application of slow sand filters is employed at iron ore Mine/Mill 1131 to further polish tailing pond effluent prior to final discharge.

Besides the application at the various facilities described above, a series of pilot-scale tests was performed at a number of facilities in the ore mining category as part of the investigation of BAT technologies described. These studies were conducted at Mine/Mill 3121, Mine/Mill/Smelter/Refinery 3107, Mill 2122 (two studies on tailing pond effluent), Smelter/Refinery 2122 (wastewater treatment plant), Mine/Mill/Smelter/Refinery 2121, Mine 3113, Mine 5102, Mill 9401, Mill 9402 (two studies) (Reference Section VIII). In each case, filtration (among other technologies) was evaluated and produced average effluent levels of TSS consistently below 10 mg/l, and usually below 5 mg/l on an average basis.

Partial Recycle

This option consists of the recycle and reuse of mill process water (not once-through mine water used as mill process water). One of the principal advantages of recycle of process water is the volume of wastewater to be treated and discharged is reduced. Although initial capital costs of installation of pumps, piping, and other equipment may be high, these are often offset by a reduction in costs associated with the treatment and discharge. Many facilities within this industry practice partial recycle including lead/zinc Mills 3105 (67 percent), 3103 (40 percent), 3101 (all needs met by recycle), gold Mill 4105 (recycle of treated water), molybdenum Mill 6102 (meets needs of mill), nickel Mill and Smelter 6106, vanadium Mill 6107, and titanium Mill 9905. In-process recycle of concentrate thickener overflow and/or filtrate produced by concentrate filtering is practiced by a number of flotation mills including 2121, 3101, 3102, 3108, 3115, 3116, 3119, 3123, and 3140. In addition, Mills 2120, 1132, 6101, and 6157 employ thickeners to reclaim water from tailings or settling ponds prior to the final discharge of these tailings to tailings ponds.

The practices described above are beneficial with respect to water conservation and recovery of metals which might be lost in the wastewater discharge. These practices are also significant with respect to wastewater treatment considerations. The in-process recycle of concentrate-thickener overflow and/or filtrate produced by concentrate filtering reduces the volume of wastewater discharged by 5 to 17 percent at mills which employ these practices. Likewise, those mills which reuse water reclaimed from tailings reduce both new water requirements and the volume discharged by 10 to 50 percent. The advantage of any practice which reduces the volume of wastewater discharged can be viewed in terms of economy of treatment and enhancement of treatment system capabilities (i.e., increased retention time of existing sedimentation basins).

The use of mine drainage as makeup in the mill is a practice that also deserves mention here as a method of reducing discharge volume to the environment. A large number of facilities in the ore mining and dressing point source category employ this practice (see Section VIII).

In general, there are four benefits resulting from adoption of this practice. They are:

1. Recovery of raw materials in processing;
2. Conservation of water;
3. Reduction of discharge to tailings ponds; and
4. Increase in performance of tailings ponds.

Implementing recycle within a facility or treatment process may require modification. Modification will be specific to each facility and each operator will have to make his own determinations.

100 Percent Recycle - Zero Discharge

This option consists of complete recycle and reuse of process water with no resulting discharge of wastewater to the environment. Many facilities in the industry have demonstrated that total recycle of process water is technically feasible. All of the iron ore mills in the Mesabi Range have demonstrated the viability of this option. Total recycle systems are also demonstrated by iron ore Mill 1105, rare earth Mill 9903 and mercury Mill 9201. Forty-six mills using froth flotation in the Copper, Lead, Zinc, Gold, Silver, Platinum and Molybdenum Ores Subcategory presently achieve zero discharge including 31 copper, five lead/zinc, five primary gold, one molybdenum and four primary silver mills.

There are two methods of water reclamation that are practiced in a number of mills. They are in-process recycle and end-of-pipe recycle. In-process recycle may involve recycle of overflow from concentrate thickeners, recycle of filtrate from concentration filters, recycle of spilled reagents or any combination of these. End-of-pipe recycle involves recycle of overflow from a tailings thickener or recycle from the tailings pond itself.

For facilities practicing mining and milling, it can be argued that in many cases the combined treatment of mine and mill wastewater is beneficial from a discharge standpoint, however, the feasibility of combining the mine and mill streams will depend on the magnitudes of:

1. The flow of mine drainage; and
2. The process water makeup flow required for the mill.

In order to achieve zero discharge at many mills, it will be necessary to treat mine water separately and use part of it as makeup water in the mill.

In comments to EPA concerning existing facilities, some facilities claimed that zero discharge by recycle would contribute to the buildup of reagents in the process water. It was further alleged that this buildup would interfere with the metallurgy of the process. To date, no data have been submitted demonstrating this contention. The cost of preventing runoff from entering tailings or sedimentation ponds and geographical locations in areas which have net precipitation are also concerns with this option for existing sources.

SELECTION AND DECISION CRITERIA

Summary of Pollutants to be Regulated

In Section VII, Selection of Pollutant Parameters, the effluent data obtained during sampling and analysis for each of the 129 toxic pollutants were reviewed by subcategory and subpart for further consideration in regulation development. In summary, all 114 of the toxic organic, and six of the toxic metal pollutants were excluded from further consideration under provisions in Paragraph 8 of the Settlement Agreement as shown below.

Toxic Pollutants		129	
Organics	-	87	(Not Detected)
Organics	-	17	(Detected at levels below EPA's nominal detection limit)
Organics	-	9	(Detected at levels too low to be effectively treated)
Organic	-	1	(Uniquely related to the facility in which it was detected)
Metals	-	6	(Detected at levels too low to be effectively treated)
		9	(Remaining for consideration)

7 Toxic Metals (arsenic, copper, lead, zinc, cadmium, mercury and nickel) Asbestos and Cyanide

The seven toxic metals, asbestos and cyanide were considered for regulation in subcategories and subparts where these pollutants were detected during sampling and analysis and were not excluded under Paragraph 8 as discussed in Section VII. Chemical oxygen demand, total iron, dissolved iron, total radium 226, dissolved radium 226, aluminum, and uranium, are regulated in the subcategories and subparts in which they were regulated under BPT.

Subcategories and Subparts in Which Toxic Pollutants Were Not Detected Are Excluded Under Paragraph 8 of the Settlement Agreement

There were subcategories and subparts in which all of the toxic pollutants were excluded from further consideration in regulation development (refer to Table VII-2, Pollutants Considered for Regulation). These include:

1. Iron ore mine drainage and mill process wastewater (not in the Mesabi Range);

2. Aluminum mine drainage;

3. Titanium mine drainage (lode ores); and

4. Titanium mines/mills employing dredging of sand deposits.

Consequently, for these subcategories and subparts, BAT effluent limitations are the same as BPT effluent limitations since there are no toxic pollutants to be controlled.

Subcategories and Subparts Which Were Not Permitted to Discharge Under BPT

No discharge of wastewater was specified for facilities in the following subcategories and subparts under BPT:

1. Iron Ore Mills in the Mesabi Range;

2. Copper, Lead, Zinc, Silver, Gold, Platinum and Molybdenum Mines and Mills that leach to recover copper;

3. Gold Mills that use cyanidation; and

4. Mercury Mills.

Facilities in these subcategories and subparts have achieved the goal of the Clean Water Act and no additional reduction of toxics is possible. Therefore, the BAT effluent limitations are the same as under BPT.

Subcategories and Subparts Where BAT Limitations Are Developed

There were subcategories and subparts in which some of the toxic pollutants were detected and are not excluded from regulation development. These include:

1. Copper, Lead, Zinc, Gold, Silver, Platinum, and Molybdenum mine drainage, mill process water from facilities using froth flotation, and placer mines;

2. Tungsten mine drainage and mill process water;

3. Mercury mine drainage;

4. Uranium mine drainage; and

5. Titanium mill process water.

Recycle was considered as an option for froth flotation mills in the copper, lead, zinc, gold, silver, platinum, and molybdenum ores subcategory. This option was rejected for froth flotation mills because of the extreme costs that would be incurred during retrofit, the downtime required to retrofit existing equipment and the impact of changing the metallurgical process.

Recycle was also considered as an option for placer mines recovering gold. The placer mining industry consists primarily of small operations located in remote areas in Alaska. Placer mining involves recovery of gold and other heavy mineral deposits by washing, dredging, or other hydraulic methods. Chemical reagents are not used in the processing of the deposits. For this reason, the pollutants of primary concern are suspended or settleable solids which may result during recovery.

Arsenic and mercury were found in placer effluents during recent studies because (1) arsenic occurs naturally in abundance in many areas of Alaska; and (2) mercury has been used extensively in the past by placer miners for recovery of gold in sluiceboxes, and mercury residuals are undoubtedly present in old deposits presently being reworked by modern day miners. Results of this study indicate that effective removal of the total suspended and settleable solids by settling also resulted in effective removal of arsenic and mercury.

At a few placer mines it may be technically feasible to recycle water for reuse in sluicing gold-bearing sediments. However, the location of most of the operations, the fact that electric power is not available to run pumps and the magnitude of the costs and energy requirements mitigate against this practice. As a result, EPA has selected settleable solids limitations based on settling ponds as the means for controlling discharges from placer mining operations. The choice of settleable solids frees the operators from having to ship samples from remote locations to laboratories for analysis. The analytical method is undemanding, inexpensive, short-term duration test that can be performed by large and small operators alike.

The settelable solids data from placer mining facilities included two separate studies of existing placer mines in Alaska and other studies performed by EPA and by departments of the State of Alaska. However, the actual data for effluent from existing settling ponds associated with gold placer mines is limited because many of the mines, including mines in the data base, have no settling facilities. Of the remaining mines which have settling ponds, it was identified that the majority, if not all, of the existing ponds for which we have data, were undersized, filled with sediment, short circuited, or otherwise poorly operated to remove settlable solids from the wastestreams of placer mines. The data from well constructed, operated, and maintained settling ponds is limited to demonstration projects and a few existing settling ponds which may not be truly representative of gold placer mining operations (e.g., mines located outside of the boundries of streams or floating dredge operations).

Cost comparisons for two treatment technologies (primary settling followed by secondary settling and primary settling with flocculation) were perfomred including the subsequent cost per ton of ore mined. However, no economic analysis was perfomred for the gold placer mining subpart because no data are available that would enable the Agency to perform cash flow analysis of placer mine operations.

Limitation for gold placer mines are reserved in the proposed BAT rulemaking in the absence of information regarding the economic impact of regulating gold placer mines and to allow the Agency to seek additional data on the effluent from well operated settling ponds assocaited with gold placer mines.

The method used to compute the achievable levels for the toxic metals is summarized below and presented in greater detail in Supplement B. The data obtained during sampling and analysis as well as that supplied by industry were reviewed and effluent levels achievable were computed for each toxic metal considered for regulatory development in Section VII. As discussed in Section VII, TSS removal technologies also remove metals, so the following TSS removal measures were considered for metal removal:

1. Secondary settling;
2. Coagulation and flocculation; and
3. Granular media filtration.

Eighteen facilities throughout the ore mining and dressing industry were identified as using multiple settling ponds; 14 facilities using coagulation and flocculation; and one facility using granular media filtration. The entire BAT and BPT data base was searched and screened to obtain 17 facilities with data. Of these 17 facilities, seven were eliminated because it was believed that they were not operated properly (e.g., short circuiting in the settling ponds was observed) or no raw (untreated) wastewater data were available to compare with treated effluent.

The facility treated effluent mean values were ranked for each of the 10 remaining facilities for each pollutant from largest to smallest. Since each facility used only one of the candidate BAT treatment technologies, the facility mean also represented a treatment technology mean value. When examining the ranked mean values, it was observed that mean values for facilities using secondary settling bracketed those for facilities using flocculation and granular media filtration. This variation indicated that the differences between facilities were greater than the differences between treatment technologies. Possibly, differences existed between the true performance capabilities of the treatment technology; however, on the basis of available data, one cannot discern such differences.

The 10 facilities were then further reduced to six by eliminating facilities whose raw (untreated) waste contained low pollutant concentrations. This was done to ensure that only those facilities which demonstrated true reduction would be included in the analysis. Data for a particular pollutant were excluded if the median raw wastewater concentration was less than the average facility effluent concentration of any other facility. Of the six facilities, five use secondary settling and one uses granular media filtration. Since there were no discernable differences in the levels achievable by the three technologies (based on available data), the least costly alternative was selected for establishing effluent limitations, secondary settling.

Achievable levels were computed by using the average of the facility averages for each pollutant to represent the average discharge. The data used were from the five facilities using secondary settling (two copper, two lead/zinc, and one silver) that remained following the screening procedures described above.

The data base indicates that within-plant effluent concentrations were approximately log normally distributed. The 30-day average maximum and daily maximum effluent limits were determined on the basis of 99th percentile estimates. The 30-day limits were determined by using the central limit theorem. The achievable levels computed for each of the metals and TSS are shown below:

	30-Day Average	Daily Maximum
Arsenic	0.01	0.05
Cadmium	0.005	0.01
Copper	0.05	0.20
Lead	0.04	0.14
Mercury	0.001	0.002
Nickel	0.10	0.40
Zinc	0.20	0.80
TSS*	10	25

*TSS limitations were computed, but TSS would be limited under BCT.

The limitations derived from the data analysis for some pollutant metals were more stringent than the BPT limitations.

Having computed achievable levels for the candidate BAT technologies, EPA then completed an environmental assessment, which analyzed the environmental significance of toxic pollutants currently discharged from facilities in this industry and also those toxic pollutants known to be discharged from this industry at BPT and expected to be discharged based on the computed achievable levels. The basis for determining the environmental significance of toxic pollutants in current discharges is a comparison of average plant effluent concentrations with Ambient Water Quality Criteria (WQC) for the protection of human health and aquatic life published by the EPA's Criteria and Standards Division (CSD) in November 1980. Because WQC for the protection of aquatic life were not developed for all of the Section 307(a)(1) toxic pollutants, the average plant effluent concentrations for pollutants lacking these WQC are compared with pollutant-specific toxicity data reported in the Ambient Water Quality Criteria Documents. The environmental significance of toxic pollutants in post-BAT discharges is determined by comparing the achievable levels with WQC or, for those pollutants lacking WQC for the protection of acquatic life, with EPA toxicity data.

Based on a review of the sampling and analysis data available for this industry, the only environmentally significant pollutants after applying the median dilution from the average receiving stream flow available (to this industry) are cadmium and arsenic. The concentration of cadmium currently being discharged from this industry (BPT) is the lowest of any industry known to discharge cadmium. In addition, the additional BAT reductions are small relative to the levels present in raw (untreated) waste streams.

In preparing the environmental assessment, the Agency also compared raw waste mass loadings to those of BPT and those expected by achievable levels. It was found that the industry's current discharge is less than 10 percent of the industry's raw waste load. This is due to the installation and proper maintenance of the Best Practicable Technology at many plants.

In other information and data available to the Agency at the time this Development Document was written, the BAT limitations proposed or being considered for metals in discharges from other industries are less stringent than those promulgated for BPT in the Ore Mining and Dressing Industry. In examining limitations for Coil Coating, Metal Finishing and Inorganic Chemicals, the following observations may be made with respect to the limitations:

1. Based on the use of similar technology (lime precipitation and settling), proposed BAT limitations on copper, cyanide, mercury and zinc for Coil Coating are less

stringent than those promulgated as BPT for Ore Mining and Dressing. Cadmium and lead limitations are more stringent.

2. Based on the use of similar technology (lime precipitation and settling), proposed BAT limitations for the following pollutants being considered for Inorganic Chemicals are less stringent than in Ore Mining BPT in the following subcategories.

- Sodium Dichromate - zinc (copper, lead, cyanide, cadmium and mercury are not regulated)

- Copper Sulfate - copper, cadmium and zinc (mercury and cyanide are not regulated; lead is more stringent)

- Nickel Sulfate - copper, cadmium, and zinc (mercury and cyanide are not regulated; lead is more stringent)

- Sodium Bisulfite - copper, mercury, and zinc (cadmium and cyanide are not regulated; lead is more stringent)

- Sodium Hydrosulfite - copper, lead and zinc (cadmium, cyanide and mercury are not regulated)

- Aluminum Fluoride - copper (cadmium, cyanide, lead, and mercury are not regulated; zinc is more stringent)

- Titanium Dioxide - cadmium, copper and zinc (cyanide and mercury are not regulated; lead is more stringent)

3. Based on the use of similar technology (lime precipitation and settling), limitations for the Common Metals Subcategory in Metal Finishing on copper and zinc are less stringent than those promulgated for Ore Mining and Dressing. Cadmium and lead limitations are more stringent.

EPA also considered the solubility of compounds in which the toxic metals are found in the industry effluent. The metals present in the wastewater from ore milling are found primarily in the ore and gangue discharged by the mill. The metals present in mine drainage similarly consist primarily of ore and gangue. The metals in ore and gangue are primarily sulfides and to a lesser extent oxides. The metals as mined generally exist as sulfides: as example, chalcopyrite ($CuFeS_2$), bornite (Cu_5FeS_4), covellite (CuS), and chalcocite (Cu_2S) for copper; galena (PbS) for lead; sphalerite (ZnS) for zinc; and argentite (Ag_2S), proustite (Ag_3AsS_3) and stephanite Ag_5S_4Sb for sliver. The metals may also exist as oxides: as example, tenorite (CuO) and cuprite (Cu_2O) for copper; pyromorphite $Pb_5Cl(PO_4)_3$ for lead; zincite (ZnO) for zinc; and wulfenite ($PbMoO_4$) for molybdenum or silicates, as example, hemimorphite, $Zn_4(Si_2O_7)(OH_2)\bullet H_2O$ and willemite (Zn_2SiO_4) for zinc (Reference 5). The oxides and silicates are less soluble than the sulfides and the sulfides are far less soluble than the hydroxides as shown in the table below.

Solublility Products of Toxic Metals

Toxic Metal	Solubility Product Metal Hydroxide	Metal Sulfide
Cadmium, Cd	13.6	26.1
Copper, Cu	18.6	35.2
Lead, Pb	16.1	26.6
Mercury, Hg	25.4	52.2
Zinc, Zn	15.7	25.2

Source: Development Document for Effluent Limitations Guidelines and Standards for the Inorganic Chemicals Manufacturing Point Source Category, EPA 440/1-79/007, June 1980.

By contrast, in many other industrial point source categories including, for example, the coal coating, metal finishing, inorganic chemical categories, toxic metals are introduced with raw waste as dissolved species. The toxic metals in those industries are present in the wastewater either in the dissolved form or as metal hydroxide. As indicated by the above table metal hydroxides are of comparatively high solubility.

The metal hydroxides (solids) present in the wastewater of these other industries have a greater potential of redissolving into solution and being available to the environment to be assimilated thereby posing a greater danger to human and aquatic species.

The metals (solids) present in the wastewater from ore milling consist primarily of the minerals associated with the ore and gangue discharged by the mill. The metals (solids) present in mine drainage similarly consists of ore and gangue. The metals in the ore and gangue are generally the sulfides of the metal and are of relatively low solubility.

After considering the environmental assessment, the BAT limitations proposed for other industries, the physical form the toxic metals, and economic factors, the Agency has concluded that nationally applicable regulations based on secondary settling or any of the other candidate BAT technologies are not warranted in the Ore Mining and Dressing Point Source Category.

Cyanide Control and Treatment

As discussed in Section VIII, cyanide compounds are used in froth flotation process of copper, lead, zinc, and molybdenum, ores. In addition, the cyanidation process is used for leaching of gold and silver ores. Consequently, residual cyanide is found in mill tailings and wastewater streams from these mills. Cyanide is also found in low concentrations in mine drainage at facilities which backfill mine stopes with the sand fraction of mill tailings.

Of the control and treatment technologies available for cyanide, consideration was given to the following options: in-process control, chemical oxidation (alkaline chlorination, hydrogen peroxide oxidation, ozonation), and natural oxidation and the incidental removal occurring in existing treatment systems (tailing ponds). These options were judged to be most applicable to the high flow volume and comparatively low concentrations of cyanide in the wastewater streams typical in this category. Another alternative which was considered was the substitution of other reagents for cyanide compounds in froth flotation processes. Bench-scale testing indicated that this alternative, although technically feasible, would require extensive testing in actual production of circumstances with specific ores. In addition, it would be difficult in these cases to predict downtime, loss of recovery (if any), and costs associated with process modifications.

Alkaline Chlorination

This method was described in detail in Section VIII, while operating cost assumptions are outlined in Section IX. Basically, oxidation of cyanide by alkaline chlorination may be accomplished by infusion of gaseous chlorine into the waste stream at a pH greater than 10, or by the addition of sodium hypochlorite (NaOCl) as an oxidant along with an alkali such as sodium hydroxide (NaOH). The alkali achieves pH adjustment and precipitation of metal hydroxides formed from the breakdown of metal-cyanide complexes.

Pilot-scale tests of alkaline chlorination treatment at Mill 6102 showed reduction of effluent cyanide concentrations from 0.19 mg/l to less than 0.1 mg/l at pH values greater than 8.8. In addition, Mill 3144 achieved reduction of effluent cyanide concentrations to an average of 0.18 mg/l from 4.72 mg/l following the installation of a full-scale alkaline-chlorination treatment system.

Ozonation

Oxidation of cyanide by ozonation is also accomplished at elevated pH (9 to 12). Copper appears to act as a catalyst in this process, which suggests that waste streams containing copper cyanide complexes may be treated more effectively by ozonation. Pilot-scale testing of ozonation at Mill 6102 showed reduction of cyanide concentration from 0.55 mg/l to less than 0.1 mg/l at pH greater than 7.4.

Hydrogen Peroxide Treatment

Hydrogen peroxide (H_2O_2) has also been tested on a limited basis as an oxidant for cyanide treatment in milling wastewater streams. This process also requires an alkaline pH and can be enhanced by a copper catalyst. Mill 6101 has achieved approximately 40 percent removal of cyanide during periods of elevated effluent levels (up to approximately 0.09 mg/l) by hydrogen peroxide oxidation.

Process Control

One characteristic of the froth flotation process which potentially affects effluent wastewater quality is the latitude available to the mill operator at the upper end of the dosage application spectrum. That is, while the addition of less than the necessary quantities of cyanide reagent may lead to loss of recovery or reduced product purity, the addition of more than the necessary quantities of cyanide reagent is not accompanied by penalties to the same degree, except of course, the cost of the additional reagent.

Close attention to mill feed characteristics and careful and frequent analysis of its mineral content can result in reduction of cyanide dosage to that actually required. In recent years, on-line analysis techniques and reagent addition controls have become available to minimize excess additions of reagent.

Few froth flotation process facilities in the industry have reported treated effluent cyanide concentrations equal to or in excess of 0.1 mg/l, and these only on an infrequent basis. Mill 6102, the largest consumer of cyanide in terms of dosage per unit of ore feed, has been observed in the past to generate effluent cyanide concentrations as high as 0.2 mg/l to 0.4 mg/l. Following installation of cyanide treatment, this facility is reporting cyanide levels less than 0.1 mg/l. Three other flotation mills have reported discharge concentrations in excess of 0.1 mg/l (2122, 3121, 6101). In each case, the cyanide dosages used in mill feed appear to be consistent with dosages reported throughout the industry and are not unusual in that respect. Fluctuations and peaking in cyanide concentrations appear to be related to short-term overdoses of cyanide in the flotation process. Few treated effluent measurements in the entire industry have exceeded 0.2 mg/l and we believe that, with close process control and reagent addition in combination with a well designed and operated treatment system, the 0.2 mg/l measurement for total cyanide can be achieved without additional treatment technology for cyanide. For the rare case where difficulty may be encountered or great reliability is required, treatment technology (i.e., chemical oxidation) is available as discussed in Section VIII and costed in Section X.

Many existing NPDES permits for ore mills contain limitations on total cyanide. As example, in EPA Region VIII, there are nine existing permits that limit total cyanide in the discharge from ore mills and these limitations vary from .02 mg/l to .2 mg/l daily maximum. In EPA Region X there are twelve permits for ore mills that limit total cyanide and these limitations vary from .01 mg/l to .3 mg/l daily maximum. Monitoring data for these permits confirm that these mills are consistently within their permit limitations on total cyanide and that the limitations can be obtained by control of the process and the incidential removal of cyanide as discussed below.

Incidental Cyanide Removal

Frequently, specific cyanide treatment technology is not necessary if close process control combined with incidental removal leads to low concentrations of total cyanide in mill water treated effluent. This incidental removal is thought to involve several mechanisms, including ultraviolet irradiation, biochemical oxidation, and natural aeration. As evidence that such mechanisms are involved, it has been noted that effluent cyanide concentrations tend to be somewhat higher during winter months when biological activity in the tailing pond is lower and ultraviolet exposure is much lower due to shortened daylight hours, less intense radiation, and ice cover on the ponds.

In addition, the association of cyanide with the depressed minerals (i.e., pyrites) will cause a portion of the cyanide to be removed together with the suspended solids and deposited in the tailing ponds.

Precision and Accuracy Study

A study of the analysis of cyanide in ore mining and processing wastewater was conducted in cooperation with the American Mining Congress to investigate the causes of analytical interferences observed and to determine what effect these interferences had on the precision of the analytical method. The purpose of this study was to evaluate the EPA-approved method and a modified method for the determination of cyanide. The modified method employed a lead acetate scrubber to remove sulfide compounds produced during the reflux-distillation step. Sulfides have been suspected of providing an interference in the colorimetric determination of cyanide concentrations. Also, several samples were spiked with thiocyanate to ascertain if this compound caused interference in the cyanide analysis.

A statistical analysis of the resultant data shows no significant difference in precision or accuracy of the two methods employed when applied to ore mining and milling wastewaters having cyanide concentrations in the 0.2 mg/l to 0.4 mg/l range. Based upon the statistical analysis, approximately 50 percent of the overall error of either method was attributed to intralaboratory error. This highlights the need for an experienced analyst to perform cyanide analyses. After considering the results of this study and the levels achieved through dose control of reagent addition in the mill, EPA considered proposing an effluent limitation of 0.2 mg/l. That limitation is based on a grab sample for any one day, and would have been subject to 100 percent error to account for the precision and accuracy of the analytical method. (See Section V above). Therefore, the Agency would have had to allow an analytical measurement of up to 0.4 mg/l. However, all the data observations in our sample were below that level. Accordingly, the Agency is proposing to exclude cyanide from national regulation in the ore mining category.

ADDITIONAL PARAGRAPH 8 EXCLUSIONS

Exclusion of Cyanide

Total cyanide is not regulated in BAT because the Agency cannot quantify a reduction in total cyanide from observed concentrations being discharged by use of technologies, known to the Administrator, Paragraph 8(a) iii of the Settlement Agreement.

The references to total cyanide levels of less than 0.2 mg/l throughout this document are for informational purposes only and are subject to the precision and accuracy of the analytical method.as discussed here and in Section V.

Exclusion of Arsenic and Nickel

EPA reviewed the achievable levels calculated based on the capabilities of the three candidate BAT treatment technologies (secondary settling, coagulation and flocculation, and granular media filtration). The Agency examined the necessity of proposing specific limitations for all seven of the toxic metals considered for regulation. Limitations on copper, lead, and zinc are necessary since these are the metals recovered from mining operations and concentrated in mills in this category. From a treatability viewpoint, control of some toxic metals (arsenic and nickel) may be achieved by limitations upon which other pollutants are controlled. As discussed in this section, since most of the metals are in suspended solids, reduction of arsenic and nickel occurs in conjunction with the removal of TSS and other toxic metals (copper, lead, and zinc).

The BAT data base for the Ore Mining and Dressing Point Source Category was searched for instances in which arsenic and nickel concentrations exceeded BPT limitations when copper, lead, and zinc concentrations were also below their respective BPT limitations. There was only one instance in over 300 samples in which a nickel or arsenic concentration exceeded their BPT limitations when BPT limitations for copper, lead, and zinc were met. The one instance was the discharge from a sedimentation pond at Facility 3103. The nickel concentration was 0.22 mg/l as opposed to the 0.20 mg/l BPT limitation.

The Agency concluded that the limitations on copper, lead, and zinc would ensure adequate control of arsenic and nickel, and under Paragraph 8(a)iii of the Settlement Agreement, arsenic and nickel are excluded from regulation.

Exclusion of Asbestos

Chrysotile asbestos was detected in wastewater samples in all subcategories and subparts within the ore mining and dressing point source category. It was detected in 90 of 91 samples throughout the entire industrial category.

EPA believes that the most appropriate way to regulate a toxic pollutant is by a direct limitation on the toxic pollutant. However, direct limitation of toxic pollutants is not always feasible. In the case of chrysotile asbestos, there is no EPA approved method of analysis for industrial wastewater samples. The method of analysis presently used was developed for drinking water samples. In addition, there are less than half a dozen laboratories in the United States that are capable of performing the analysis by this method.

Chrysotile asbestos is known to be present in many ore deposits throughout the country (Reference 6). As ore is mined and subsequently milled, it is subjected to a variety of crushing and size reduction operations. As a result, smaller solids are formed, the chrysotile asbestiform fibers are liberated as the small solids are made, and end up in mine drainage and mill process water.

The possibility of the chrysotile asbestos fibers being present in waste streams for the same reasons and in the same relative proportions as the solids, led to the examination of the EGD sampling data in an attempt to establish a relationship between chrysotile asbestos and TSS.

Review of analyses for asbestos and TSS in samples of untreated and treated wastewater shows that as TSS is reduced by treatment, observed asbestos concentrations are also reduced.

Intake water samples (from 26 industrial categories) and POTW effluents were reviewed to get an indication of background levels for chrysotile asbestos. The values of chrysotile asbestos ranged from 3.5×10^4 (detection limit) to 1.63×10^8 with a mean value of 1.1×10^7 fibers per liter. The treated waste stream sample values for chrysotile asbestos in the ore data ranged from 10^4 (detection limit) to 10^8 fibers per liter.

The Agency has determined that when TSS is reduced to the BPT effluent limitations for ore mining and dressing, observed chrysotile asbestos levels are reduced to or below background levels. Therefore, EPA is excluding chrysotile asbestos from regulation since it is effectively controlled by technologies upon which TSS limitations are based, Paragraph 8(a)iii of the Settlement Agreement.

Exclusion of Pollutants Detected in a Single Source and Uniquely Related to That Source

There are 19 operating uranium mills in the United States, 18 of which now achieve zero discharge of process wastewater. There are no uranium mills that commingle process wastewater with mine drainage and it is anticipated that none of these zero discharge mills would elect to treat and discharge at the BPT limitations because of the expense to install technology required, i.e., ion exchange, ammonia stripping, lime precipitation, barium chloride coprecipitation, and settling.

EPA is excluding uranium mills from BAT, since there is only one discharging facility and it is believed that none of the other existing facilities will commingle mine drainage and mill process wastewater. Uranium mills are not regulated in BAT because the pollutants found in the discharge are uniquely related to this single source, Paragraph 8(a)iii of the Settlement Agreement.

SECTION XI

BEST CONVENTIONAL POLLUTANT CONTROL TECHNOLOGY

Section 301(b)(2)(E) of The Act requires that there be achieved, not later than July 1, 1984, effluent limitations for categories and classes of point sources, other than publicly-owned treatment works, that require the application of the best conventional pollutant control technology (BCT) for control of conventional pollutants as identified in Section 304(a)(4). The pollutants that have been defined as conventional by the Agency, at this time, are biochemical oxygen demand, suspended solids, fecal coliform, oil and grease, and pH.

BCT is not an additional limitation; rather, it replaces BPT for the control of conventional pollutants. BCT must be evaluated for cost effectiveness and a comparison made between the cost and level of reduction of conventional pollutants from the discharge of publicly owned treatment works (POTW) and the cost and level of reduction of such pollutants from a class or category of industrial sources.

The technologies considered for treatment of conventional pollutants are the same as those considered for treatment of toxic pollutants. As discussed in Section X, the Agency has determined that the BAT limitations for toxics are equivalent to BPT limitations for the ore mining and dressing industry. BCT limitations for the conventional pollutants, TSS, and pH are also proposed at BPT levels. Accordingly, by definition, BCT for this industry meets any BCT cost test because there is no incremental cost to remove conventional pollutants beyond BPT.

Summary of Best Conventional Technology

Pollutant	(mg/l)	Daily Maximum (mg/l)	30-Day Average
TSS		30	20
pH		Within the range of 6 to 9	

The Agency is currently developing a new BCT cost methodology. It is possible, though unlikely, that a treatment technology more stringent than BPT will provide additional removal of conventional pollutants and pass the new cost test, when developed. In that event, the Agency will propose new BCT limitations.

SECTION XII

NEW SOURCE PERFORMANCE STANDARDS (NSPS)

The basis for new source performance standards (NSPS) under Section 306 of the Act is through application of the best available demonstrated technology. New facilities have the opportunity to implement the best and most efficient ore mining and milling processes and wastewater technologies. Congress, therefore, directed EPA to consider the best demonstrated process changes and end-of-pipe treatment technologies capable of reducing pollution to the maximum extent feasible.

GENERAL PROVISIONS

Several items of discussion apply to options in more than one subcategory. To avoid repetition, these items are discussed here.

Relief From No Discharge Requirement

For new sources that must achieve no discharge of process wastewater, the Agency provides relief upon the occurrence of a 10-year, 24-hour precipitation event or snow melt of equivalent volume. Excess water resulting from the occurrence shall not be subject to effluent limitations.

Relief from no discharge of process wastewater is also granted for those facilities located in net precipitation areas and is the same relief granted to BAT limitations requiring no discharge.

Relief From Effluent Limitations for Those Facilities Permitted to Discharge

The relief is exactly the same as that granted for existing sources under BAT. Excess water resulting from precipitation from a facility designed, constructed, and maintained to contain or treat the maximum volume of process wastewater discharged during any 24-hour period, including the volume that would result from a 10-year, 24-hour precipitation event or snow melt of equivalent volume, shall not be subject to the limitations set forth in 40 CFR 440.

Commingling Provisions

For new sources that combine for treatment waste streams from various sources, the quantity and quality of each pollutant or pollutant property in the combined discharge that is subject to effluent limitations shall not exceed the quantity and quality of each pollutant or pollutant property that would have been discharged had each waste stream been treated separately. The discharge flow from a combined discharge shall not exceed the

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volume that would have been discharged had each waste stream been treated separately.

NSPS OPTIONS CONSIDERED

The Agency considered the following NSPS options:

Option One. Require achievement of performance standards in each subcategory based on the same technology as BAT (NSPS = BAT).

Option Two. Require standards based on a complete water recycle system (NSPS = zero discharge).

NSPS SELECTION AND DECISION CRITERIA

EPA has selected performance standards based on the same technology as BAT for all facilities in the ore mining and dressing point source category, except those facilities using froth flotation in the copper, lead, zinc, gold, silver, platinum, and molybdenum subcategory and mills in the uranium subcategory.

Subcategories and Subparts Under Option 1

Option 1 (NSPS = BAT) has been selected for iron ore mills in the Mesabi range; copper, lead, zinc, silver, gold, platinum, and molybdenum mills that use leaching to recover copper and the cyanidation process for the recovery of gold; and mercury mills since BAT specifies zero discharge. Option 1 (NSPS = BAT) has also been selected for iron ore mine drainage, iron ore mills, aluminum mine drainage, copper, lead, zinc, gold, silver, platinum, and molybdenum mine drainage, titanium mine drainage, dredges and mills, and mercury mine drainage. The concentration levels of toxic metals found in new sources in these subcategories and subparts are expected to be similar to existing sources. Following the implementation of BAT, toxic metals will be found at or near detection levels or at concentrations below the practical limits of additional technology. Further reduction of these pollutants can not be technically or economically justified.

Subcategories and Subparts Under Option 2 (NSPS = zero discharge)

EPA is proposing that new source froth flotation mills achieve zero discharge of process wastewater. EPA considered zero discharge based on recycle for existing copper, lead, zinc, gold, silver, platinum, and molybdenum mills using froth flotation, but rejected it because of the extensive retrofit required at some existing facilities, the cost of retrofitting, and the possible changes required in the process. This concern does not apply to new sources. Zero discharge is a demonstrated technology at 46 of the 90 froth flotation mills for which EPA has data (see wastewater discharges as summarized in Tables IX-2 through IX-10) and meets the definition of standard of performance permitting zero discharge of pollutants. New sources have the option to recycle because the metallurgical process can be adjusted and designed to recycle process wastewater before the actual construction of the new source. While reagent buildup has been mentioned by industry as a potential problem in extractive metallurgy, no evidence has been submitted to validate ths assertion.

There are new sources anticipated in copper, lead, zinc, gold, silver, and molybdenum mining. Standards applied to these new source waste streams should reflect the best treatment levels achievable by the froth flotation segment of the industry.

A study of existing froth flotation mills reveals that a large percentage of these facilities are effectively achieving 100 percent recycle of mill water. Many of the facilities practicing 100 percent recycle are located in arid regions, but some facilities are located in humid regions. A summary of some existing facilities follows.

Copper Ore

Of the 35 known froth flotation copper mills in the United States, 31 achieve zero discharge of process wastewater.

Lead/Zinc Ores

Five of the 27 active froth flotation mills in the lead/zinc subcategory achieve zero discharge.

Gold

Four of the five primary gold facilities employing froth flotation techniques discharge process wastewater.

Silver

Three of the four known primary silver facilities which use froth flotation methods are achieving zero discharge of process wastewater.

Molybdenum

Of the three molybdenum operations employing the froth flotation process, one facility achieves zero discharge of recycle.

In Section IX (Table IX-3 through Table IX-10) cost comparisons for treatment technologies are made for existing sources. It is believed that new source mills will have similar mill capacity, water use per ton of ore, and pollutants in the raw wastewater as existing mills. Therefore, costs of technology for new sources will approach those costs determined for existing sources. The cost of 100% recycle for mills is generally less than the cost determined for pH adjustment of the wastewater (lime addition). Assuming the same settling pond, or tailings pond, would be required for recycle as for pH adjustment and settling, the cost to recycle at a new source is approximately the same, or less, than the cost to implement the technology (pH adjustment and settle) upon which BPT and BAT effluent limitations are based.

EPA is proposing that new source uranium mills achieve zero discharge of process wastewater. For this subpart, EPA considered zero discharge for BAT based on total impoundment and evaporation, or recycle and reuse of the mill process water, or a combination of these technologies. Because the pollutants detected in the current discharge from this subpart are uniquely related to one point source, the single mill discharging, the uranium mill subpart is excluded from BAT under Paragraph 8 authority of the Settlement Agreement (see Section X).

However, the Agency believes that for new sources a standard of performance must be proposed. Otherwise, additional discharges (new sources) could occur that obviously would not be unique to one source. New source uranium mills are anticipated by the Agency. New mill capacity is anticipated to replace existing mills and to maintain the current production of uranium oxide as lower grade ore must be mined. Also, an increase in demand for uranium oxide is anticipated that will require new mills. Uranium mills can achieve zero discharge as indicated by the fact that 18 of 19 existing mills currently achieve no discharge.

As discussed in Section X, ammonia stripping, lime precipitation, barrium chloride co-precipitation or ion exchange, and settling are the identified technologies for uranium mills to meet BPT limitations on metals, radium 226, ammonia and TSS. The cost to implement the technologies to meet the BPT limitations for a new uranium mill is more than the cost to implement recycle or evaporation ponds (or the combination of the two) to meet a no discharge requirement.

SECTION XIII

PRETREATMENT STANDARDS

Section 307(b) of the Act requires EPA to promulgate pretreatment standards for both existing sources (PSES) and new sources (PSNS) of pollution which discharge their wastes into publicly owned treatment works (POTWs). These pretreatment standards are designed to prevent the discharge of pollutants which pass through, interfere with, or are otherwise incompatible with the operation of POTWs. In addition, the Clean Water Act of 1977 adds a new dimension of these standards by requiring pretreatment of pollutants, such as heavy metals, that limit POTW sludge management alternatives. The legislative history of the Act indicates that pretreatment standards are to be technology based and, with respect to toxic pollutants, analogous to BAT. The Agency has promulgated general pretreatment regulations which establish a framework for the implementation of these statutory requirements (see 43 FR 27736, 16 June 1978).

EPA is not proposing pretreatment standards for existing sources (PSES) or new sources (PSNS) in the ore mining and dressing point source category at this time nor does it intend to promulgate such standards in the future since there are no known or anticipated discharges to publicly owned treatment works (POTWs).

SECTION XIV

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Hanna Mining Company
Hecla Mining Company
Homestake Mining Company
Idarado Mining Company
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Rawhide Mining Company
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SECTION XVI

GLOSSARY

absorption: The process by which a liquid is drawn into and tends to fill permeable pores in a porous solid body; also the increase in weight of a porous solid body resulting from the penetration of liquid into its permeable pores.

acid copper: Copper electrode deposited from an acid solution of a copper salt, usually copper sulfate.

acid cure: In uranium extraction, sulfation of moist ore before leach.

acid leach: (a) Metallurgical process for dissolution of values by means of acid solution (used on the sandstone ores of low lime content); (b) In the copper industry, a technology employed to recover copper from low grade ores and mine dump materials when oxide (or mixed oxide-sulfide, or low grade sulfide) mineralization is present, by dissolving the copper minerals with either sulfuric acid or sulfuric acid containing ferric iron. Four methods of leaching are employed: dump, heap, in-situ, and vat (see appropriate definitions).

acid mine water: (a) Mine water which contains free sulfuric acid, mainly due to the weathering of iron pyrites; (b) Where sulfide minerals break down under the chemical influence of oxygen and water, the mine water becomes acidic and can corrode ironwork.

activator, activating agent: A substance which when added to a mineral pulp promotes flotation in the presence of a collecting agent. It may be used to increase the floatability of a mineral in a froth, or to reflect a depressed (sunk mineral).

adit: (a) A horizontal or nearly horizontal passage driven from the surface for the working or dewatering of a mine; (b) A passage driven into a mine from the side of a hill.

adsorption: The adherence of dissolved, colloidal, or finely divided solids on the surface of solids with which they are brought into contact.

aeroflocs: Synthetic water-soluble polymers used as flocculating agents.

all sliming: (a) Crushing all the ore in a mill to so fine a state that only a small percentage will fail to pass through a 200-mesh screen; (b) Term used for treatment of gold ore which is ground to a size sufficiently fine for agitation as a cyanide pulp, as opposed to division into coarse sands for static leaching and fine slimes for agitation.

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alluminothermic process: The reduction of oxides in an exothermic reaction with finely divided aluminum.

alluvial deposit; placer deposit: Earth, sand, gravel or other rock or mineral materials transported by and laid down by flowing water. Alluvial deposits generally take the form of (1) surface deposits; (2) river deposits; (3) deep leads; and (4) shore deposits.

alunite: A basic potassium aluminum sulfate, $KAl_3(OH)_6(SO_4)2$. Closely resembles kaolinite and occurs in similar locations.

amalgamation: The process by which mercury is alloyed with some other metal to produce amalgam. It was used extensively at one time for the extraction of gold and silver from pulverized ores, now is largely superseded by the cyanide process.

AN-FO - Ammonium nitrate: Fuel oil blasting agents.

asbestos minerals: Certain minerals which have a fibrous structure, are heat resistant, chemically inert and possessing high electrical insulating qualities. The two main groups are serpentine and amphiboles. Chrysotile (fibrous serpentine, $3MgO . 2SiO_2 . 2H_2O$) is the principal commercial variety. Other commercial varieties are amosite, crocidolite, actinolite, anthophyllite, and tremolite.

azurite: A blue carbonate of copper, $Cu_3(CO_3)2(OH)_2$, crystallizing in the monoclinic system. Found as an alteration product of chalcopyrite and other sulfide ores of copper in the upper oxidized zones of mineral veins.

bastnasite; bastnaesite: A greasy, wax-yellow to reddish-brown weakly radioactive mineral, $(Ce,La)(CO_3)F$, most commonly found in contact zones, less often in pegmatites.

bauxite: (a) A rock composed of aluminum hydroxides, essentially $Al_2O_3 . 2H_2O$. The principal ore of aluminum; also used collectively for lateritic aluminous ores. (b) Composed of aluminum hydroxides and impurities in the form of free silica, clay, silt, and iron hydroxides. The primary minerals found in such deposits are boehmite, gibbsite, and diaspore.

Bayer Process: Process in which impure aluminum in bauxite is dissolved in a hot, strong, alkalai solution (normally NaOH) to form sodium aluminate. Upon dilution and cooling, the solution hydrolyzes and forms a precipitate of aluminum hydroxide.

bed: The smallest division of a stratified series and marked by a more or less well-defined divisional plane from the materials above and below.

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beneficiation: (a) The dressing or processing of ores for the purpose of (1) regulating the size of a desired product, (2) removing unwanted constituents, and (3) improving the quality, purity, assay grade of a desired product; (b) Concentration or other preparation of ore for smelting by drying, flotation, or magnetic separation.

Best Available Technology Economically Achievable (BAT): The level of technology applicable to effluent limitations to be achieved by 1 July 1983, for industrial discharges to surface waters as defined by Section 301(b)(1)(A) of the Act.

Best Practicable Control Technology Currently Available (BPT): The level of technology applicable to effluent limitations to be achieved by 1 July 1977, for industrial discharges to surface waters as defined by Section 301(b)(1)(A) of the Act.

byproduct: A secondary or additional product.

carbon absorption: A process utilizing the efficient absorption characteristics of activated carbon to remove both dissolved and suspended substances.

carnotite: A bright yellow uranium mineral, $K_2(UO_2)_2(VO_4)_2 \cdot 3H_2O$.

cationic collectors: In flotation, amines and related organic compounds capable of producing positively charged hydrocarbon-bearing ions for the purpose of floating miscellaneous minerals, especially silicates.

cationic reagents: In flotation, surface active substances which have the active constituent in the positive ion. Used to flocculate and to collect minerals that are not flocculated by the reagents, such as oleic acid or soaps, in which the surface-active ingredient is the negative ion.

cement copper: Copper precipitated by iron from copper sulfate solutions.

cerium metals: Any of a group of rare-earth metals separable as a group from other metals occurring with them and in addition to cerium includes lanthanum, praseodymium, neodymium, promethium, samarium and sometimes europium.

cerium minerals: Rare earths; the important one is monazite.

chalcocite: Copper sulfide, Cu_2S.

chalcopyrite: A sulfide of copper and iron, $CuFeS_2$.

chert: Cryptocrystalline silica, distinguished from flint by flat fracture, as opposed to conchoidal fracture.

chromite: Chrome iron ore, $FeCr_2O_4$.

chrysocolla: Hydrated copper silicate, $CuSiO_3 . 2H_2O$.

chrysotile: A metamorphic mineral, an asbestos, the fibrous variety of serpentine. A silicate of magnesium, with silica tetrahedra arranged in sheets.

cinnabar: Mercury sulfide, HgS.

claim: The portion of mining ground held under the Federal and local laws by one claimant or association, by virtue of one location and record. A claim is sometimes called a "location."

clarification: (a) The cleaning of dirty or turbid liquids by the removal of suspended and colloidal matter; (b) The concentration and removal of solids from circulating water in order to reduce the suspended solids to a minimum; (c) In the leaching process, usually from pregnant solution, e.g., gold-rich cyanide prior to precipitation.

classifier: (a) A machine or device for separating the constituents of a material according to relative sizes and densities thus facilitating concentration and treatment. Classifiers may be hydraulic or surface-current box classifiers. Classifiers are also used to separate sand from slime, water from sand, and water from slime; (b) The term classifier is used in particular where an upward current of water is used to remove fine particles from coarser material; (c) In mineral dressing, the classifier is a device that takes the ball-mill discharge and separates it into two portions--the finished product which is ground as fine as desired, and oversize material.

coagulation: The binding of individual particles to form flocs or agglomerates and thus increase their rate of settlement in water or other liquid (see also flocculate).

coagulator: A soluble substance, such as lime, which when added to a suspension of very fine solid particles in water causes these particles to adhere in clusters which will settle easily. Used to assist in reclaiming water used in flotation.

collector: A heteropolar compound containing a hydrogen-carbon group and an ionizing group, chosen for the ability to adsorb selectively in froth flotation processes and render the adsorbing surface relatively hydrophobic. A promoter.

columbite; tantalite; niobite: A natural oxide of niobium (columbium), tantalum, ferrous iron, and manganese, found in granites and pegmatites, $(Fe, Mn) Nb, Ta)_2O_6$.

concentrate: (a) In mining, the product of concentration; (b) To separate ore or metal from its containing rock or earth; (c) The enriched ore after removal of waste in a beneficiation mill, the clean product recovered in froth flotation.

concentration: Separation and accumulation of economic minerals from gangue.

concentrator: (a) A plant where ore is separated into values (concentrates) and rejects (tails). An appliance in such a plant, e.g., flotation cell, jig, electromagnet, shaking table. Also called mill; (b) An apparatus in which, by the aid of water or air and specific gravity, mechanical concentration of ores is performed.

conditioners: Those substances added to the pulp to maintain the proper pH to protect such salts as NaCN, which would decompose in an acid circuit, etc. Na_2CO_3 and CaO are the most common conditioners.

conditioning: Stage of froth-flotation process in which the surfaces of the mineral species present in a pulp are treated with appropriate chemicals to influence their reaction when the pulp is aerated.

copper minerals: Those of the oxidized zone of copper deposits (zone of oxidized enrichment) include azurite, chrysocolla, copper metal, cuprite, and malachite. Those of the underlying zone (that of secondary sulfide enrichment) include bornite, chalcocite, chalcopyrite, covellite. The zone of primary sulfides (relatively low in grade) includes the unaltered minerals bornite and chalcopyrite.

crusher: A machine for crushing rock or other materials. Among the various types of crushers are the ball-mill, gyratory crusher, Hadsel mill, hammer mill, jaw crusher, rod mill, rolls, stamp mill, and tube mill.

cuprite: A secondary copper mineral, Cu_2O.

cyanidation: A process of extracting gold and silver as cyanide slimes from their ores by treatment with dilute solutions of potassium cyanide and sodium cyanide.

cyanidation vat: A large tank, with a filter bottom, in which sands are treated with sodium cyanide solution to dissolve out gold.

cyclone: (a) The conical-shaped apparatus used in dust collecting operations and fine grinding applications; (b) A classifying (or concentrating) separator into which pulp is fed, so as to take a circular path. Coarser and heavier fractions of solids report at the apex of long cone while finer particles overflow from central vortex.

daughter: Decay product formed when another element undergoes radioactive disintegration.

decant structure: Apparatus for removing clarified water from the surface layers of tailings or settling ponds. Commonly used structure include decant towers in which surface waters flow over a gate (adjustable in height) and down the tower to a conduit generally buried beneath the tailings, decant weirs over which water flows to a channel external to the tailings pond, and floating decant barges which pump surface water out of the pond.

dense-media separation: (a) Heavy media separation, or sink float. Separation of heavy sinking from light floating mineral particles in a fluid of intermediate density; (b) Separation of relatively light (floats) and heavy ore particles (sinks), by immersion in a bath of intermediate density.

Denver cell: A flotation cell of the subaeration type, in wide use. Design modifications include receded-disk, conical-disk, and multibladed impellers, low-pressure air attachments, and special froth withdrawal arrangements.

Denver jig: Pulsion-suction diaphragm jig for fine material, in which makeup (hydraulic) water is admitted through a rotary valve adjustable as to portion of jigging cycle over which controlled addition is made.

deposit: Mineral or ore deposit is used to designate a natural occurrence of a useful mineral or an ore, in sufficient extent and degree of concentration to invite exploitation.

depressing agent; depressor: In the froth floation process, a substance which reacts with the particle surface to render it less prone to stay in the froth, thus causing it to wet down as a tailing product (contrary to activator).

detergents, synthetic: Materials which have a cleansing action like soap but are not derived directly from fats and oils. Used in ore flotation.

development work: Work undertaken to open up ore bodies as distinguished from the work of actual ore extraction or exploratory work.

dewater: To remove water from a mine usually by pumping, drainage or evaporation.

differential flotation: Separating a complex ore into two or more valuable minerals and gangue by flotation; also called selective flotation. This type of flotation is made possible by the use of suitable depressors and activators.

discharge: Outflow from a pump, drill hole, piping system, channel, weir or other discernible, confined or discrete conveyance (see also point source).

dispersing agent: Reagent added to flotation circuits to prevent flocculation, especially of objectionable colloidal slimes. Sodium silicate is frequently added for this purpose.

dredge; dredging: A large floating contrivance for underwater excavation of materials using either a chain of buckets, suction pumps, or other devices to elevate and wash alluvial deposits and gravel for gold, tin, platinum, heavy minerals, etc.

dressing: Originally referred to the picking, sorting, and washing of ores preparatory to reduction. The term now includes more elaborate processes of milling and concentration of ores.

drift mining: A term applied to working alluvial deposits by underground methods of mining. The paystreak is reached through an adit or a shallow shaft. Wheelbarrows or small cars may be used for transporting the gravel to a sluice on the surface.

dump leaching: Term applied to dissolving and recovering minerals from subore-grade materials from a mine dump. The dump is irrigated with water, sometimes acidified, which percolates into and through the dump, and runoff from the bottom of the dump is collected, and a mineral in solution is recovered by chemical reaction. Often used to extract copper from low grade, waste material of mixed oxide and sulfide mineralization produced in open pit mining.

effluent: The wastewater discharged from a point source to navigable waters.

electrowinning: Recovery of a metal from an ore by means of electrochemical processes, i.e., deposition of a metal on an electrode by passing electric current through an electrolyte.

eluate: Solutions resulting from regeneration (elution) of ion exchange resins.

eluent: A solution used to extract collected ions from an ion exchange resin or solvent and return the resin to its active state.

exploration: Location of the presence of economic deposits and establishing their nature, shape, and grade and the investigation may be divided into (1) preliminary, and (2) final.

extraction: (a) The process of mining and removal of ore from a mine. (b) The separation of a metal or valuable mineral from an ore or concentrate. (c) Used in relation to all processes that are used in obtaining metals from their ores. Broadly, these processes involve the breaking down of the ore both mechanically (crushing) and chemically (decomposition), and the separation of the metal from the associated gangue.

ferruginous: Containing iron.

ferruginous chert: A sedimentary deposit consisting of chalcedony or of fine-grained quartz and variable amounts of hematite, magnetite, or limonite.

ferruginous deposit: A sedimentary rock containing enough iron to justify exploitation as iron ore. The iron is present, in different cases, in silicate, carbonate, or oxide form, occurring as the minerals chamosite, thuringite, siderite, hematite, limonite, etc.

flask: A unit of measurement for mercury; 76 pounds.

flocculant: An agent that induces or promotes flocculation or produces floccules or other aggregate formation, especially in clays and soils.

flocculate: To cause to aggregate or to coalesce into small lumps or loose clusters, e.g., the calcium ion tends to flocculate clays.

flocculating agent; flocculant: A substance which produces flocculation.

flotation: The method of mineral separation in which a froth created in water by a variety of reagents floats some finely crushed minerals, whereas other minerals sink.

flotation agent: A substance or chemical which alters the surface tension of water or which makes it froth easily. The reagents used in the flotation process include pH regulators, slime dispersants, resurfacing agents, wetting agents, conditioning agents, collectors, and frothers.

friable: Easy to break, or crumbling naturally.

froth, foam: In the flotation process, a collection of bubbles resulting from agitation, the bubbles being the agency for raising (floating) the particles of ore to the surface of the cell.

frother(s): Substances used in flotation processes to make air bubbles sufficiently permanent principally by reducing surface tension. Common frothers are pine oil, creyslic acid, and amyl alcohol.

gangue: Undesirable minerals associated with ore.

glory hole: A funnel-shaped excavation, the bottom of which is connected to a raise driven from an underground haulage level or is connected through a horizontal tunnel (drift) by which ore may also be conveyed.

gravity separation: Treatment of mineral particles which exploits differences between their specific gravities. Their sizes and shapes also play a minor part in separation. Performed by means of jigs, classifiers, hydrocyclones, dense media, shaking tables, Humphreys spirals, sluices, vanners and briddles.

grinding: (a) Size reduction into relatively fine particles. (b) Arbitrarily divided into dry grinding performed on mineral containing only moisture as mined, and wet grinding, usually done in rod, ball or pebble mills with added water.

heap leaching: A process used in the recovery of copper from weathered ore and material from mine dumps. The liquor seeping through the beds is led to tanks, where it is treated with scrap iron to precipitate the copper from solution. This process can also be applied to the sodium sulfide leaching of mercury ores.

heavy-media separation: See dense-media separation.

hematite: One of the most common ores of iron, Fe_2O_3, which when pure contains about 70% metallic iron and 30% oxygen. Most of the iron produced in North America comes from the iron ranges of the Lake Superior District, especially the Mesabi Range, Minnesota. The hydrated variety of this ore is called limonite.

Huntington-Heberlein Process: A sink-float process employing a galena medium and utilizing froth flotation as the means of medium recovery.

hydraulic mining: (a) Mining by washing sand and soil away with water which leaves the desired mineral. (b) The process by which a bank of gold-bearing earth and rock is excavated by a jet of water, discharged through the converging nozzle of a pipe under great pressure. The debris is carried away with the same water and discharged on lower levels into watercourses below.

hydrolysate; hydrolyzate: A sediment consisting partly of chemically undecomposed, finely ground rock powder and partly of insoluble matter derived from hydrolytic decomposition during weathering.

hydrometallurgy: The treatment of ores, concentrates, and other metal-bearing materials by wet processes, usually involving the solution of some component, and its subsequent recovery from the solution.

ilmenite: An iron-black mineral, FeO . TiO_2. Resembles magnetite in appearance but is readily distinguished by feeble magnetic character.

in-situ leach: Leaching of broken ore in the subsurface as it occurs, usually in abandoned underground mines which previously employed block-caving mining methods.

ion(ic) exchange: The replacement of ions on the surface, or sometimes within the lattice, of materials such as clay.

iron formation: Sedimentary, low grade, iron ore bodies consisting mainly of chert and fine-grained quartz and ferric oxide segregated in bands or sheets irregularly mingled (see also taconite).

jaw crusher: A primary crusher designed to reduce large rocks or ores to sizes capable of being handled by any of the secondary crushers.

jig: A machine in which the feed is stratified in water by means of a pulsating motion and from which the stratified products are separately removed, the pulsating motion being usually obtained by alternate upward and downward currents of the water.

jigging: (a) The separation of the heavy fractions of an ore from the light fractions by means of a jig. (b) Up and down motion of a mass of particles in water by means of pulsion.

laterite: Red residual soil developed in humid, tropical, and subtropical regions of good drainage. It is leached of silica and contains concentrations particularly of iron oxides and hydroxides and aluminum hydroxides. It may be an ore of iron, aluminum, manganese, or nickel.

launder: (a) A trough, channel, or gutter usually of wood, by which water is conveyed; specifically in mining, a chute or trough for conveying powdered ore, or for carrying water to or from the crushing apparatus. (b) A flume.

leaching: (a) The removal in solution of the more soluble minerals by percolating waters. (b) Extracting a soluble metallic compound from an ore by selectively dissolving it in a suitable solvent, such as water, sulfuric acid, hydrochloric acid, etc. The solvent is usually recovered by precipitation of the metal or by other methods.

leach ion-exchange flotation process: A mixed method of extraction developed for treatment of copper ores not amenable to direct flotation. The metal is dissolved by leaching, for example, with sulfuric acid, in the presence of an ion exchange resin. The resin recaptures the dissolved metal and is then recovered in a mineralized froth by the flotation process.

leach precipitation float: A mixed method of chemical reaction plus flotation developed for such copper ores as chrysocolla and the oxidized minerals. The value is dissolved by leaching with acid, and the copper is reprecipitated on finely divided particles of iron, which are then recovered by flotation, yielding an impure concentrate in which metallic copper predominates.

lead minerals: The most important industrial one is galena (PbS), which is usually argentiferous. In the upper parts of deposits the mineral may be altered by oxidation to cerussite ($PbCO_3$) or anglesite ($PbSO_4$). Usually galena occurs in intimate association with sphalerite (ZnS).

leucoxene: A brown, green, or black variety of sphene or titanite, $CaTiSiO$, occurring as monoclinic crystals. An earthy alteration product consisting in most instances of rutile; used in the production of titanium tetrachloride.

lime: Quicklime (calcium oxide) obtained by calcining limestone or other forms of calcium carbonate. Loosely used for hydrated lime (calcium hydroxide) and incorrectly used for pulverized or ground calcium carbonate in agricultural lime and for calcium in such expressions as carbonate of lime, chloride of lime, and lime feldspar.

lime slurry: A form of calcium hydroxide in aqueous suspension that contains considerable free water.

limonite: Hydrous ferric oxide $FeO(OH) \cdot nH_2O$. An important ore of iron, occurring in stalactitic, mammillary, or earthy forms of a dark brown color, and as a yellowish-brown powder. The chief constituent of bog iron ore.

liquid-liquid extraction, solvent extraction: A process in which one or more components are removed from a liquid measure by intimate contact with a second liquid, which is itself nearly insoluble in the first liquid and dissolves the impurities and not the substance that is to be purified.

lode: A tabular deposit of valuable mineral between definite boundaries. Lode, as used by miners, is nearly synonymous with the term vein as employed by geologists.

magnetic separation: The separation of magnetic materials from nonmagnetic materials using a magnet. An important process in the beneficiation of iron ores in which the magnetic mineral is separated from nonmagnetic material, e.g., magnetite from other minerals, roasted pyrite from sphalerite.

magnetic separator: A device used to separate magnetic from less magnetic or nonmagnetic materials. The crushed material is conveyed on a belt past a magnet.

magnetite, magnetic iron ore: Natural black oxide of iron, Fe_3O_4. As black sand, magnetite occurs in placer deposits, and also as lenticular bands. Magnetite is used widely as a suspension solid in dense-medium washing of coal and ores.

malachite: A green, basic cupric carbonate, $Cu_2(OH)2CO_3$, crystallizing in the monoclinic system. It is a common ore of copper and occurs typically in the oxidation zone of copper deposits.

manganese minerals: Those in principal production are pyrolusite, some psilomelane, and wad (impure mixture of manganese and other oxides).

manganese nodules: The concretions, primarily of manganese salts, covering extensive areas of the ocean floor. They have a layer configuration and may prove to be an important source of manganese.

manganese ore: A term used by the Bureau of Mines for ore containing 35 percent or more manganese and may include concentrate, nodules, or synthetic ore.

manganiferous iron ore: A term used by the Bureau of Mines for ores containing 5 to 10 percent manganese.

manganiferous ore: A term used by the Bureau of Mines for any ore of importance for its manganese content containing less than 35 percent manganese but not less than 5 percent manganese.

mercury minerals: The main source is cinnabar, HgS.

mill: (a) Reducing plant where ore is concentrated and/or metals recovered. (b) Today the term has been broadened to cover the whole mineral treatment plant in which crushing, wet grinding, and further treatment of the ore is conducted. (c) In mineral processing, one machine, or a group, used in comminution.

minable: (a) Capable of being mined. (b) Material that can be mined under present day mining technology and economics.

mine: (a) An opening or excavation in the earth for the purpose of excavating minerals, metal ores or other substances by digging. (b) A word for the excavation of minerals by means of pits, shafts, levels, tunnels, etc., as opposed to a quarry, where the whole excavation is open. In general the existence of a mine is determined by the mode in which the mineral is obtained, and not by its chemical or geologic character. (c) An excavation beneath the surface of the ground from which mineral matter of value is extracted. Excavations for the extraction of ore or other economic minerals not requiring work beneath the surface are designated by a modifying word or phrase as: (1) opencut mine - an excavation for removing minerals which is open to the weather; (2) steam shovel mine - an opencut mine in which steam shovels or other power shovels are used for loading cars; (3) strip mine - a stripping, an openpit mine in which the overburden is removed from the exploited material before the material is taken out; (4) placer mine - a deposit of sand, gravel or talus from which some valuable mineral is extracted; and (4) hydraulic mine - a placer mine worked by means of a stream of water directed against a bank of sand, gravel, or talus. Mines are commonly known by the mineral or metal extracted, e.g., bauxite mines, copper mines, silver mines, etc. (d) Loosely, the word mine is used to mean any place from which minerals are extracted, or ground which it is hoped may be mineral bearing. (e) The Federal and State courts have held that the word mine, in statutes reserving mineral lands, included only those containing valuable mineral deposits. Discovery of a mine: In statutes relating to mines the word discovery is used: (1) In the sense of uncovering or disclosing to view ore or mineral; (2) of finding out or bringing to the knowledge the existence of ore, or mineral, or other useful products which were unknown; and (3) of exploration, that is, the more exact blocking out or ascertainment of a deposit that has already been discovered. In this sense it is practically synonymous with development, and has been so used in the U.S. Revenue Act of 19 February 1919 (Sec. 214, subdiv. A10, and Sec. 234, subdiv. A9) in allowing depletion of mines, oil and gas wells. Article 219 of Income and War Excess Profits Tax Regulations No. 45, construes discovery of a mine as: (1) The bona fide discovery of a commercially valuable deposit of ore or mineral, of a value materially in excess of the cost of discovery in natural exposure or by drilling or other exploration conducted above or below the ground; and (2) the development and proving of a mineral or ore deposit which has been apparently worked out to be a mineable deposit or ore, or mineral having a value in excess of the cost of improving or development.

mine drainage: (a) Mine drainage usually implies gravity flow of water to a point remote from mining operation. (b) The process of removing surplus ground or surface water by artificial means.

mineral: An inorganic substance occurring in nature, though not necessarily of inorganic origin, which has (1) a definite chemical composition, or commonly a characteristic range of chemical composition, and (2) distinctive physical properties, or molecular structure. With few exceptions, such as opal (amorphous) and mercury (liquid), minerals are crystalline solids.

mineral processing; ore dressing; mineral dressing: The dry and wet crushing and grinding of ore or other mineral-bearing products for the purpose of raising concentrate grade; removal of waste and unwanted or deleterious substances from an otherwise useful product; separation into distinct species of mixed minerals; chemical attack and dissolution of selected values.

modifier(s): (a) In froth flotation, reagents used to control alkalinity and to eliminate harmful effects of colloidal material and soluble salts. (b) Chemicals which increase the specific attraction between collector agents and particle surfaces, or conversely which increase the wettability of those surfaces.

molybdenite: The most common ore of molybdenum, MoS_2.

molybdenite concentrate: Commercial molybdenite ore after the first processing operations. Contains about 90% MoS_2 along with quartz, feldspar, water, and processing oil.

monazite: A phosphate of the cerium metals and the principal ore of the rare earths and thorium. Monoclinic. One of the chief sources of thorium used in the manufacture of gas mantles. It is a moderately to strongly radioactive mineral, $(Ce, La, Y, Th)PO_4$. It occurs widely disseminated as an accessory mineral in granitic igneous rocks and gneissic metamorphic rocks. Detrital sands in regions of such rocks may contain commercial quantities of monazite. Thorium-free monazite is rare.

New Source Performance Standard (NSPS): Performance standards for the industry and applicable new sources as defined by Section 306 of the Act.

niccolite: A copper-red arsenide of nickel which usually contains a little iron, cobalt, and sulfur. It is one of the chief ores of metallic nickel.

nickel minerals: The nickel-iron sulfide, pentlandite (Fe, Ni) is the principal present economic source of nickel, and garnierite (nickelmagnesium hydrosilicate) is next in economic importance.

oleic acid: A mono-saturated fatty acid, $CH_3(CH_2)_{\_}CH{:}CH(CH_2)_7$ COOH. A common component of almost all naturally occurring fats as well as tall oil. Most commercial oleic acid is derived from animal tallow or natural vegetable oils.

open-pit mining, open cut mining: A form of operation designed to extract minerals that lie near the surface. Waste, or overburden, is first removed, and the mineral is broken and loaded. Important chiefly in the mining of ores of iron and copper.

ore: (a) A natural mineral compound of the elements of which one at least is a metal. Applied more loosely to all metalliferous rock, though it contains the metal in a free state, and occasionally to the compounds of nonmetallic substances, such as sulfur. (b) A mineral of sufficient value as to quality and quantity which may be mined with profit.

ore dressing: The cleaning of ore by essentially physical means and the removal of certain valueless portion. Synonym for concentration. The same as mineral dressing.

ore reserve: The term usually restricted to ore of which the grade and tonnage have been established with reasonable assurance by drilling and other means.

oxidized ores: The alteration of metalliferous minerals by weathering and the action of surface waters, and the conversion of the minerals into oxides, carbonates, or sulfates.

oxidized zone: That portion of an ore body near the surface, which has been leached by percolating water carrying oxygen, carbon dioxide or other gases.

pegmatite: An igneous rock of coarse grain size usually found as a crosscutting structure in a larger igneous mass of finer grain size.

pelletizing: A method in which finely divided material is rolled in a drum or on an inclined disk, so that the particles cling together and roll up into small, spherical pellets.

pH modifiers: Proper functioning of a cationic or anionic flotation reagent is dependent on the close control of pH. Modifying agents used are soda ash, sodium hydroxide, sodium silicate, sodium phosphates, lime, sulfuric acid, and hydrofluoric acid.

placer mine: (a) A deposit of sand, gravel, or talus from which some valuable mineral is extracted. (b) To mine gold, platinum, tin or other valuable minerals by washing the sand, gravel, etc.

placer mining: The extraction of heavy mineral from a placer deposit by concentration in running water. It includes ground sluicing, panning, shoveling gravel into a sluice, scraping by power scraper, excavation by dragline or extraction by means of various types of dredging activities.

platinum minerals: Platinum, ruthenium rhodium, palladium, osmium, and iridium are members of a group characterized by high specific gravity, unusual resistance to oxidizing and acidic attack, and high melting point.

point source: Any discernible, confined and discrete conveyance, including but not limited to any pipe, ditch, channel, tunnel, conduit, well, discrete fissure, container, rolling stock, concentrated animal feeding operation, or vessel or other floating craft, from which pollutants are or may be discharged.

pregnant solution: A value bearing solution in a hydrometallurgical operation.

pregnant solvent: In solvent extraction, the value-bearing solvent produced in the solvent extraction circuit.

promoter: A reagent used in froth-flotation process, usually called the collector.

rare-earth deposits: Sources of cerium, terbium, yttrium, and related elements of the rare-earth's group, as well as thorium.

raw mine drainage: Untreated or unprocessed water drained, pumped or siphoned from a mine.

reagent: A chemical or solution used to produce a desired chemical reaction; a substance used in assaying or in flotation.

reclamation: The procedures by which a disturbed area can be reworked to make it productive, useful, or aesthetically pleasing.

recovery: A general term to designate the valuable constituents of an ore which are obtained by metallurgical treatment.

reduction plant: A mill or a treatment place for the extraction of values from ore.

roast: To heat to a point somewhat short of fuzing in order to expel volatile matter or effect oxidation.

rougher cell: Flotation cells in which the bulk of the gangue is removed from the ore.

roughing: Upgrading of run-of-mill feed either to produce a low grade preliminary concentrate or to reject valueless tailings at an early stage. Performed by gravity on roughing tables, or in flotation in a rougher circuit.

rutile: Titanium dioxide, TiO_2.

scintillation counter: An instrument used for the location of radioactive ore such as uranium. It uses a transparent crystal which gives off a flash of light when struck by a gamma ray, and a photomultiplier tube which produces an electrical impulse when the light from the crystal strikes it.

selective flotation: See differential flotation.

settling pond: A pond, natural or artificial, for recovering solids from an effluent.

siderite: An iron carbonate, $FeCO_3$.

slime, slimes: A material of extremely fine particle size encountered in ore treatment.

sludge: The precipitant or settled material from a wastewater.

slurry: (a) Any finely divided solid which has settled out as from thickeners. (b) A thin watery suspension.

solvent extraction: See liquid-liquid extraciton.

sphalerite: Zinc sulfide, ZnS.

stibnite: An antimony sulfide, Sb_2S_3. The most important ore of antimony.

suction dredge: (a) Essentially a centrifugal pump mounted on a barge. (b) A dredge in which the material is lifted by pumping through a suction pipe.

sulfide zone: That part of a lode or vein not yet oxidized by the air or surface water and containing sulfide minerals.

surface active agent: One which modifies physical, electrical, or chemical characteristics of the surface of solids and also surface tensions of solids or liquid. Used in froth flotation (see also depressing agent, flotation agent).

tabling: Separation of two materials of different densities by passing a dilute suspension over a slightly inclined table having a reciprocal horizontal motion or shake with a slow forward motion and a fast return.

taconite: (a) The cherty or jaspery rock that encloses the Mesabi iron ores in Minnesota. In a somewhat more general sense, it designates any bedded ferruginous chert of the Lake Superior District. (b) In Minnesota practice, is any grade of extremely hard, lean iron ore that has its iron either in banded or well-desseminated form and which may be hematite or magnetite, or a combination of the two within the same ore body (Bureau of Mines).

taconite ore: A type of highly abrasive iron ore now extensively mined in the United States.

tailing pond: Area closed at lower end by constraining wall or dam to which mill effluents are run.

tailings: (a) The parts, or a part, of any incoherent or fluid material separated as refuse, or separately treated as inferior in quality or value; leavings; remainders; dregs. (b) The gangue and other refuse material resulting from the washing, concentration, or treatment of ground ore. (c) Those portions of washed ore that are regarded as too poor to be treated further; used especially of the debris from stamp mills or other ore dressing machinery, as distinguished from concentrates.

tall oil: The oily mixture of rosin acids, and other materials obtained by acid treatment of the alkaline liquors from the digesting (pulping) of pine wood. Used in drying oils, in cutting oils, emulsifiers, and in flotation agents.

tantalite: A tantalate of iron and manganese (Fe, Mn) Ta_2O, crystallizing in the orthorhombic system.

tetrahedrite: A mineral, the part with Sb greater than As of the tetrahedrite-tenantite series, $Cu_3(Sb, As)S_3$. Silver, zinc, iron and mercury may replace part of the copper. An important ore of copper and silver.

thickener: A vessel or apparatus for reducing the amount of water in a pulp.

thickening: (a) The process of concentrating a relatively dilute slime pulp into a thick pulp, that is, one containing a smaller percentage of moisture, by rejecting liquid that is essentially solid free. (b) The concentration of the solids in a suspension with a view to recovering one fraction with a higher concentration of solids than in the original suspension.

tin minerals: Virtually all the industrial supply comes from cassiterite (SnO_2), though some has been obtained from the sulfide minerals stannite, cylindrite, and frankeite. The bulk of cassiterite comes from alluvial workings.

titanium minerals: The main commercial minerals are rutile (TiO_2) and ilmenite ($FeTiO_3$).

tyuyamunite: A yellow uranium mineral $(Ca(UO_2)_2VO_4)_2 . 3H_2O$. It is the calcium analogue of carnotite.

uraninite: Essentially UO_2. It is a complex uranium mineral containing also rare earths, radium, lead, helium, nitrogen and other elements.

uranium minerals: More than 150 uranium bearing minerals are known to exist, but only a few are common. The five primary uranium-ore minerals are pitchblende, uraninite, davidite, coffinite, and brannerite. These were formed by deep-seated hot solutions and are most commonly found in veins or pegmatites. The secondary uranium-ore minerals, altered from the primary minerals by weathering or other natural processes, are carnotite, tyuyamunite and metatyuyamunite (both are very similar to carnotite), torbernite and metatorbernite, autunite and meta-autunite, and uranophane.

vanadium minerals: Those most exploited for industrial use are patronite (VS_4), roscoelite (vanadium mica), vanadinite ($Pb\_Cl(VO_4)_3$), carnotite and chlorovanadinite.

vat leach: Employs the dissolution of copper oxide minerals by sulfuric acid from crushed, non-porous ore material placed in confined tanks. The leach cycle is rapid and measured in days.

weir: An obstruction placed across a stream for the purpose of diverting the water so as to make it flow through a desired channel, which may be an opening or notch in the weir itself.

wetting agent: A substance that lowers the surface tension of water and thus enables it to mix more readily. Also called surface active agent.

Wilfley table: Widely used for of shaking table. A plane rectangle is mounted horizontally and can be sloped about its long axis. It is covered with linoleum (occasionally rubber) and has longitudinal riffles dying at the discharge end to a smooth cleaning area, triangular in the upper corner. Gentle and rapid throwing motion is used on the table longitudinally. Sands, usually classified for size range are fed continuously and worked along the table with the aid of feedwater, and across riffles downslope by gravity tilt adjustment, and added washwater. At the discharge end, the sands have separated into bands, the heaviest and smallest uppermost, the lightest and largest lowest.

xanthate: Common specific promoter used in flotation of sulfide ores. A salt or ester of xanthic acid which is made of an alcohol, carbon disulfite and an alkalai.

xenotime: A yttrium phosphate, YPO_4, often containing small quantities of cerium, terbium, and thorium, closely resembling zircon in crystal form and general appearance.

yellow cake: (a) A term applied to certain uranium concentrates produced by mills. It is the final precipitate formed in the milling process. It is usually considered to be ammonium diuranate, $(NH_4)_2U_2O_7$, or sodium diuranate, $Na_2U_2O_7$, but the composition is variable and depends upon the precipitating conditions. (b) A common form of triuranium octoxide, U_3O_8, is yellow cake, which is the powder obtained by evaporating an ammonia solution of the oxide.

zinc minerals: The main source of zinc is sphalerite (ZnS), but some smithsonite, hemimorphite, zincite, willemite, and franklinite are mined.

zircon: A mineral, $ZrSiO_4$. The chief ore of zirconium.

zircon, rutile, ilmenite, monazite: A group of heavy minerals which are usually considered together because of their occurence as black sand in natural beach and dune concentration.

APPENDIX A

INDUSTRY PROCESSES
(Refer to Section III)

The following ore types are included: iron, copper, lead-zinc, gold, silver, molybdenum, tungsten, vanadium, mercury, uranium, antimony and titanium.

Iron Ore Milling Processes

Beneficiation of iron ore includes such operations as crushing, screening, blending, grinding, concentrating, classifying, briquetting, sintering and agglomerating. Beneficiation is often done at or near the mine site. Methods selected are based on physical and chemical properties of the crude ore. General techniques utilized in the beneficiation of iron ore are illustrated in Figure A-1. Processes enhance either the chemical or physical characteristics of the crude ore to make more desirable feed for the blast furnace. Beneficiation methods have been developed to upgrade 20 to 30 percent iron 'taconite' ores into high-grade materials.

Physical concentrating processes, such as washing, remove unwanted sand, clay, or rock from crushed or screened ore. For those ores not amenable to simple washing operations, other physical methods are used such as jigging, heavy-media separation, flotation, and magnetic separation. Jigging involves stratification of ore and gangue by utilizing pulsating water currents. Heavy-media separation employs water suspension of ferrosilicon whereby iron ore particles sink while the majority of gangue (quartz, etc.) floats. The flotation process uses air bubbles attached to iron particles conditioned with flotation reagents to separate iron from the gangue. Magnetic separation techniques are used on ores containing magnetite.

At the present time, there are only three iron ore flotation plants in the United States. Figure A-2 illustrates a typical flowsheet used in an iron ore flotation circuit, while Table A-1 lists types and amounts of flotation reagents used per ton of ore processed. Various flotation methods which utilize these reagents are listed in Table A-2. The most commonly adopted flowsheet for the beneficiation of low grade magnetic taconite ores is illustrated in Figure A-3. Low grade ores containing magnetite are very susceptible to concentrating processes, yielding a high quality blast furnace feed. Higher grade iron ores containing hematite cannot be upgraded much above 55 percent iron. Figure A-4 illustrates the beneficiation of a fine-grained hematite ore.

Agglomeration, which follows concentration processes, increases the particle size of iron ore and reduces "fines" which normally would be lost in the flue gases. Agglomerating methods include sintering, pelletizing, briquetting, and nodulizing. Sintering involves the mixing of small portions of coke and limestone with the iron ore, followed by combustion. A granular, coarse, porous product is formed. Pelletizing involves the formation of pellets or balls composed of iron ore fines, followed by heating (Figure A-5 illustrates a typical pelletizing operation). Hot ore briquetting requires no binder, is less sensitive to changes in feed composition, requires little or no grinding and requires less fuel than sintering. Small or large lumps of regular shape are formed. Nodules or lumps (nodulizing) are formed when ores are charged into a rotary kiln and heated to incipient fusion temperatures.

Copper Ore Milling Processes

Processing of copper ores may involve hydrometallurgical or physical-chemical separation from the gangue material. A general scheme of methods employed for recovery of copper from ores is shown in Figure A-6. These methods include dump, heap, vat and in-situ leaching, and froth flotation.

Cement copper is produced from dump, heap and in-situ leaching and cathode copper is produced by electrowinning the pregnant solution from a vat leach. Major copper areas employing dump, heap and in-situ leaching are shown in Figure A-7.

Copper bearing froth from the froth flotation process is thickened, filtered and sent to a smelter whereby blister copper (98 percent Cu) is produced. The blister copper is then sent to a refinery which produces pure copper (99.88 to 99.9 percent Cu) for market.

One combination of the hydrometallurgical and physical-chemical processes, termed LPF (leach-precipitation-flotation) has enabled the copper industry to process oxide and sulfide minerals efficiently. Also, tailings from the vat leaching process, if they contain significant sulfide copper, can be sent to the flotation circuit to float copper sulfide, while the vat leach solution undergoes iron precipitation or electrowinning to recover copper dissolved from oxide ores by acid.

Lead-Zinc Ore Milling Processes

Generally, lead-zinc ores are not of high enough grade to be smelted directly, therefore it is sent through the milling process first. In most cases, the only process utilized is froth flotation, but in some cases, preliminary gravity separation is practiced prior to flotation. The general milling procedure is to crush the ore and then grind it, in a closed circuit with rod mills, ball mills and classifying equipment, to a small enough size to allow the ore minerals to be freed from the gangue. Chemical reagents are then added which, in the presence of forced air bubbles, produce selective flotation and separation of the desired ore minerals. In some cases, the reagents used in the flotation process are added in the mill; in other cases, the fine material from the mill flows to a conditioner (mixing tank), where the reagents are added. The particular reagents utilized are a function of the mineral concentrates to be recovered. The specific choice of reagents used at a facility is usually the result of determining empirically which reagents yield the optimum mineral values versus reagent costs. In general, lead and zinc as well as copper sulfide flotations are run at elevated pH (8.5 to 11, generally) levels so that frequent pH adjustments with hydrated lime ($CaOH_2$) are common. Other reagents commonly used are:

Reagent	Purpose
Methyl Isobutyl-carbinol	Frother
Propylene Glycol Methyl Ether	Frother
Long-Chain Aliphatic Alcohols	Frother
Pine Oil	Frother
Potassium Amyl Xanthate	Collector
Sodium Isopropol Xanthate	Collector
Sodium Ethyl Xanthate	Collector
Dixanthogen	Collector
Isopropyl Ethyl Thionocarbonate	Collectors
Sodium Diethyl-dithiophosphate	Collectors
Zinc Sulfate	Zinc Depressant
Sodium Cyanide	Zinc Depressant
Copper Sulfate	Zinc Activant
Sodium Dichromate	Lead Depressant
Sulfur Dioxide	Lead Depressant
Starch	Lead Depressant
Lime	pH Adjustment

The finely ground ore slurry is introduced into a series of flotation cells, where the slurry is agitated and air is introduced. The desired minerals are rendered hydrophobic (non-water-accepting) by surface coating with appropriate reagents. Usually, several cells are operated in a countercurrent flow pattern, with the final concentrate being floated off the last cell (cleaner) and the tails being removed from the first or rougher cells.

In many cases, more than one mineral is recovered. In such cases, differential flotation is practiced. The flow diagram in Figure A-8 depicts a typical differential flotation process for recovery of lead and zinc sulfides. Chemicals which induce hydrophilic (affinity-for-water) behavior by surface interaction are added to prevent one of the minerals from floating in the initial separation. The underflow of tailings from this separation is then treated with a chemical which overcomes the depressing effect and allows the flotation of the other mineral.

The floated concentrates are dewatered (usually by thickening and filtration), and the final concentrate--which contains some residual water--is eventually shipped to a smelter for metal recovery. The liquid overflow from the concentrate thickeners is typically recycled in the mill.

After the recovery of the desirable minerals, a large volume of tailings or gangue material remains as underflow from the last rougher cell in the flow scheme. These tails are typically adjusted to a slurry suitable for hydraulic transport to the treatment facility, i.e., tailing pond. In some cases, the coarse tailings are removed by a cyclone separator and then pumped in to the mine for backfilling.

The tailings from a lead/zinc flotation mill contain residual solids from the original ore which has been finely ground to allow mineral recovery. The tailings also contain dissolved solids and excess mill reagents. In cases where the mineral content of the ore varies, excess reagents will undoubtedly be present when the ore grade drops suddenly, conversely lead and zinc will escape with the tails if high-grade ore creates a reagent-starved system. Accidental spilling of the chemical reagents used are another source of adverse discharges from a mill.

Figure A-8 depicts a typical lead-zinc ore mining and processing operation.

Gold Ore Milling Processes

Milling practices applicable to the processing and recovery of gold and gold-containing ores are cyanidation, amalgamation, flotation, and gravity concentration. All of these processes have been used in the beneficiation of ore mined from lode deposits. Placer operations, however, employ only gravity methods which in the past were sometimes used in conjunction with amalgamation.

Prior to 1970, the amalgamation process was used to recover nearly 1/4 of the gold produced domestically. Since that time, environmental concerns have caused restricted use of mercury. As a result, the percent of gold produced which was recovered by the amalgamation process dropped from 20.3 percent in 1970 to 0.3 percent in 1972. At the same time, the use of cyanidation processes was increasing. In 1970, 36.7 percent of the gold produced domestically was recovered by cyanidation, and this increased to 54.6 percent in 1972.

The amalgamation process as currently practiced (used by a single mill in Colorado) involves crushing and grinding of the lode ore, gravity separation of the gold-bearing black sands by jigging, and final concentration of the gold by batch amalgamation of the sands in a barrel amalgamator. In the past, amalgamation of lode ore has been performed in either the grinding mill, on plates, or in special amalgamators. Placer gold/silver-bearing gravels are beneficiated by gravity methods, and, in the past, the precious metal-bearing sands generally were batch amalgamated in barrel amalgamators. However, amalgamation in specially designed sluice boxes were also practiced.

There are basically four methods of cyanidation currently being used in the United States: heap leaching, vat leaching, agitation leaching, and the recently developed carbon-in-pulp process. Heap leaching is a process used primarily for the recovery of gold from low-grade ores. This is an inexpensive process and, as a result, has also been used recently to recover gold from old mine waste dumps. Higher grade ores are often crushed, ground, and vat leached or agitated/leached to recover the gold.

In vat leaching, a vat is filled with the ground ore (sands) slurry, water is allowed to drain off, and the sands are leached from the top with cyanide, which solubilizes the gold (Figure A-9). Pregnant cyanide solution is collected from the bottom of the vat and sent to a holding tank. In agitation leaching, the cyanide solution is added to a ground ore pulp in thickeners, and the mixture is agitated until solution of the gold is achieved (Figure A-10). The cyanide solution is collected by decanting from the thickeners.

Cyanidation of slimes, generated during wet grinding, is currently being done by a recently developed process, carbon-in-pulp (Figure A-9). The slimes are mixed with a cyanide solution in large tanks, and the solubilized gold cyanide is collected by adsorption onto activated charcoal. Gold is stripped from the charcoal using a small volume of hot caustic; an electrowinning process is used for final recovery of the gold in the mill. Bullion is subsequently produced at a refinery.

Gold in the pregnant cyanide solutions from heap, vat, or agitate leaching processes is recovered by precipitation with zinc dust. The precipitate is collected in a filter press and sent to a smelter for the production of bullion.

Recovery of gold by flotation processes is limited, and less than 3 percent of the gold produced in 1972 was recovered in this manner. This method employs a froth flotation process to float and collect the gold-containing minerals (Figure A-11). The one operation that uses this method, further processes tailings from the flotation circuit by the agitation/cyanidation method to recover the residual gold values.

Gold has historically been recovered from placer gravels by purely physical means. Present practice involves gravity separation, which is normally accomplished in a sluice box. Typically, a sluice box consists of an open box in which a simple rectangular sluice plate is mounted on a downward incline. To effect the separation of gold from gravel or sand, a perforated metal sheet is fitted on the bottom of the loading box and riffle structures are mounted on the bottom of the sluice plate. These riffles may consist of wooden strips or steel or plastic plates which are angled away from the direction of flow in a manner designed to create pockets and eddy currents for the collection and retention of gold.

During actual sluicing operations, pay gravels (i.e., goldbearing gravels) are loaded into the upper end of the sluice box and washed down the sluice plate with water, which enters at right angles to (or against the direction of) gravel feed. Density differences allow the particles of gold to settle and become entrapped in the spaces between the riffle structures, while the less-dense gravel and sands are washed down the sluice plate. Eddy currents keep the spaces between riffle structures free of sand and gravel but are not strong enough to wash out the gold which collects there.

Other types of equipment which may be employed in physical separation operations include jigs, tables, and screens. However, this equipment is typically found only at dredging operations.

Cleanup of gold recovered by gravity methods is normally accomplished with small (102 cm (40 in.)) sluices, screens, and finally, by hand-picking impurities from the gold.

Silver Ore Milling Processes

Present extractive metallurgy for silver was developed over a period of more than 100 years. Initially, silver, as the major product, was recovered from rich oxidized ores by relatively crude methods. As the ores became leaner and more complex, an improved extractive technology was developed. Today, silver production is predominantly as a byproduct, and is largely related to the production of lead, zinc, and copper from the processing of sulfide ores by froth flotation and smelting. Free-milling, easily liberated gold/silver ores, processed by amalgamation and cyanidation, now contribute only 1 percent of the domestic silver produced. Primary sulfide ores, processed by flotation and smelting, account for 99 percent.

Selective froth flotation processing can effectively and efficiently beneficiate almost any type and grade of sulfide ore. This process employs various well-developed reagent combinations and conditions to enable the selective recovery of many different sulfide minerals in separate concentrates of high quality. The reagents commonly used in the process are generally classified as collectors, promotors, modifiers, depressants, activators, and frothing agents. Essentially, these reagents are used in combination to cause the desired sulfide mineral to float and be collected in a froth while the undesired minerals and gangue sink. Practically all the ores presently milled require fine grinding to liberate the sulfide minerals from one another and from the gangue minerals.

A circuit which exemplifies the current practice of froth flotation for the primary recovery of silver from silver ores or complex ores is shown in Figure A-12. Primary recovery of silver occurs mainly from the mineral tetrahedrite, $(Cu, Fe, Zn, Ag)_{12}Sb_4S_{13}$. A tetrahedrite concentrate contains approximately 25 to 32 percent copper in addition to the 25.72 to 44.58 kilograms per metric ton (750 to 1,300 troy ounces per ton) of silver. A low-grade (3.43 kg per metric ton; 100 troy ounces per ton) silver/pyrite concentrate is produced at one mill. Antimony may comprise up to 18 percent of the tetrahedrite concentrate and may or may not be extracted prior to shipment to a smelter.

Various other silver-containing minerals are recovered as byproducts of primary copper, lead, and/or zinc operations. Where this occurs, the usual practice is to ultimately recover the silver from the base-metal flotation concentrates at the smelter or refinery.

Molybdenum Ore Milling Processes

The only commercially important ore of molybdenum is molybdenite, MoS_2. It is universally concentrated by flotation. Significant quantities of molybdenite concentrate are recovered as a byproduct in the milling of copper and tungsten ores.

Flotation concentration has become a mainstay of the ore milling industry. Because it is adaptable to very fine particle sizes (less than 0.01 mm, or 0.0004 inch), it allows high rates of recovery from slimes which are inevitably generated in crushing and grinding and are not generally amenable to physical processing. As a physicochemical surface phenomenon, it can often be made highly specific, allowing production of high-grade concentrates from very-low-grade ore. Its specificity also allows separation of different ore minerals (e.g., CuS and MoS_2) where desired, and operation with minimum reagent consumption since reagent interaction is typically only with the particular materials to be floated or depressed.

The major operating plants in the industry recover molybdenite by flotation. Vapor oil is used as the collector, and pine oil is used as a frother. Lime is used to control pH of the mill feed and to maintain an alkaline circuit. In addition, Nokes reagent and sodium cyanide are used to prevent flotation of galena and pyrite with the molybdenite. A generalized, simplified flowsheet for an operation recovering only molybdenite is shown in Figure A-13. Water use in this operation currently amounts to approximately 1.8 tons of water per ton of ore processed, essentially all of which is process water. Reclaimed water from thickeners at the mill site (shown on the flowsheet) amounts to only 10 percent of total use.

Where byproducts are recovered with molybdenite, a somewhat more complex mill flowsheet results, although the molybdenite recovery circuits remain quite similar. A very simplified flow diagram for such an operation is shown in Figure A-14. Pyrite flotation and monazite flotation are accomplished at acid pH (4.5 and 1.5, respectively), thereby increasing the likelihood of solubilizing heavy metals. Flow volumes at those locations in the circuit are low, however, and neutralization occurs upon combination with the main mill water flows for delivery to the tailing ponds. Water flow for this operation amounts to approximately 2.3 tons per ton of ore processed, nearly all of which is process water in contact with the ore. Essentially 100 percent recycle of mill water from the tailing ponds at this mill is prompted by limited water availability as well as by environmental considerations.

Tungsten Ore Milling Processes

Commercially important tungsten ores include the scheelite ($CaWO_4$) and wolframite series, wolframite ($((Fe, MN)\ WO_4)$), ferberite ($FeWO_4$), and huebnerite ($MnWO_4$). Concentration is by a wide variety of techniques. Gravity concentration, by jigging, tabling, or sink/float methods, is frequently employed. Because sliming due to the high friability of scheelite ore (most U.S. ore is scheelite) reduces recovery by gravity techniques, fatty-acid flotation may be used to increase recovery. Leaching may also be employed as a major beneficiation step and is frequently practiced to lower the phosphorus content of concentrates. Ore generally contains about 0.6 percent tungsten, and concentrates containing about 70 percent WO_3 are produced. A tungsten concentrate is also produced as a byproduct of molybdenum milling at one operation in a process involving gravity separation, flotation, and magnetic separation.

Figure A-15 depicts a simplified flow diagram for a small tungsten concentrator.

Vanadium Ore Processes

Eighty-six percent of vanadium oxide production has recently been used in the preparation of ferrovanadium. Although a fair share of U.S. vanadium production is derived as a byproduct of the mining of uranium, there are other sources of vanadium ores. The environmental considerations at mine/mill operations not involving radioactive constituents are fundamentally different from environmental considerations important to uranium operations, and it seems appropriate to consider the former operation separately. Vanadium is considered as part of this industry segment: (a) because of the similarity of nonradioactive vanadium recovery operations to the processes used for other ferroalloy metals and (b) because, in particular, hydrometallurgical processes like those used in vanadium recovery are becoming more popular in SIC 1061.

Vanadium is chemically similar to columbium (niobium) and tantalum, and ores of these metals may be beneficiated in the same type of process used for vanadium. There is also some similarity to tungsten, molybdenum, and chromium.

Recovery of vanadium phosphate rocks in Idaho, Montana, Wyoming, and Utah--which contain about 28 percent P_2O_5, 0.25 percent V_2O_5, and some Cr, Ni, and Mo--yields vanadium as a byproduct of phosphate fertilizer production. Ferrophosphate is first prepared by smelting a charge of phosphate rock, silica, coke, and iron ore (if not enough iron is present in the ore). The product when separated from the slag typically contains 60 percent iron, 25 percent phosphorus, 3 to 5 percent chromium, and 1 percent nickel. It is pulverized, mixed with soda ash (Na_2CO_3) and salt, and roasted at 750 to 800 degrees Celsius (1382 to 1472 degrees Fahrenheit). Phosphorus, vanadium, and chromium are converted to water-soluble trisodium phosphate, sodium metavanadate, and sodium chromate, while the iron remains in insoluble form and is not extracted in a water leach following the roast.

Phosphate values are removed from the leach in three stages of crystallization. Vanadium can be recovered as V_2O_5 (redcake) by acidification, and chromium is precipitated as lead chromate. By this process, 85 percent of the vanadium, 65 percent of the chromium, and 91 percent of the phosphorus can be extracted.

Another, basically non-radioactive, vanadium ore, with a grade of 1 percent V_2O_5, is found in a vanidiferous, mixed-layer montmorillonite/illite and geothite/montroseite matrix. This ore is recovered by salt roasting, following extrusion of pellets, to yield sodium metavanadate, which is concentrated by solvent extraction. Slightly soluble ammonium vanadate is precipitated from the stripping solution and calcined to yield vanadium pentoxide. A flow chart for this process is shown in Figure A-16.

Mercury Ore Milling Processes

The principal mineral source of mercury is cinnabar (HgS). The domestic industry has been centered in California, Nevada, and Oregon. Mercury has also been recovered from ore in Arizona, Alaska, Idaho, Texas, and Washington and is recovered as a byproduct from gold ore in Nevada and zinc ore in New York.

Until recently, the typical practice of the industry has been to feed mercury ore directly into rotary kilns for recovery of mercury by roasting. This has been such an efficient method that extensive beneficiation is precluded. However, with the depletion of high grade ores, concentration of low-grade mercury ores is becoming more important. The ore may be crushed and sometimes screened to provide a feed suitable for furnacing. Gravity concentration is also done in a few cases, but its use is limited since mercury minerals crush more easily and more finely than gangue rock.

Flotation is the most efficient method for beneficiating mercury ores when beneficiation is practiced. An advantage of flotation, especially for low-grade material, is the high ratio of concentration that results. This permits proportionate reductions in the size and costs of the final mercury extraction process. Only recently has flotation of mercury been practiced in the United States. During 1975, a single mill, located in Nevada, began operation to beneficiate mercury ore by this method (Figure A-17). The concentrate produced is furnaced at the same facility to recover elemental mercury. The ore, which averages 4.8 kg of mercury per metric ton (9.5 lb/short ton), is obtained from a nearby open-pit mine; the major ore minerals present are cinnabar (HgS) and corderoite ($Hg_3S_2Cl_2$).

Uranium Ore Milling Processes

Blending, Crushing and Roasting. Ore from the mine can be quite variable in consistency and grade. Procedures have been developed to weigh and radiometrically assay the ores. This is done to achieve uniform grade and consistency.

Ore high in vanadium is sometimes roasted with sodium chloride after crushing. This converts insoluble heavy-metal vanadates (vanadium complex) and carnotite to more soluble sodium vanadate, which is then extracted with water. Ores high in organics may be roasted to carbonize and oxidize the organics and prevent clogging of hydrometallurgical processes. Clay bearing ores attain improved filtering and settling characteristics by roasting at 300 degrees Celsius (572 degrees Fahrenheit).

Grinding. Ore is ground less than 0.6 mm (28 mesh) (0.024 in.) for acid leaching, less than 0.7 mm (200 mesh) for alkaline leaching in rod or ball mills using water (or preferably, leach) to obtain a pulp density of about two-thirds solids. Screw classifiers, thickeners, or cyclones are sometimes used to control size or pulp density.

Acid Leach. Ores with a calcium carbonate ($CaCO_3$) content of less than 12 percent are preferentially leached in sulfuric acid, which extracts values quickly (in four hours to a day), and at a lower capital and energy cost than an alkaline leach. Any tetravalent uranium must be oxidized to the uranyl form by adding an oxidizing agent (typically, sodium chlorate or manganese dioxide), which is believed to facilitate the oxidation of U(4) to U(6) in conjunction with the reduction of Fe (3) to Fe (2) at a redox (reduction/oxidation) potential of about minus 450 mV. Free-acid concentration is held to between 1 and 100 grams per liter. The larger concentrations are suitable when vanadium is to be extracted. The reactions taking place in acid oxidation and leaching are:

$2UO_2 + O_2$ --- $2UO_3$

$2UO_3 + 2H_2SO_4 + 5H_2O$ --- $2(UO_2SO_4) \cdot 7H_2O$

Uranyl sulfate (UO_2SO_4) forms a complex, hydrouranyl trisulfuric acid ($H_4UO_2(SO_4)_3$ in the leach, and the anions of this acid are extracted for value.

Alkaline Leach. A solution of sodium carbonate (40 to 50 g per liter) in an oxidizing environment selectively leaches uranium and vanadium values from their ores. The values may be precipitated directly from the leach by raising the pH and adding sodium hydroxide. The supernatant can be recycled after its exposure to carbon dioxide. A controlled amount of sodium bicarbonate (10 to 20 g per liter) is added to the leach to lower pH which prevents spontaneous precipitation.

This leaching process is slower than acid leaching since other ore components are not attached and these ore components tend to shield the uranium values. Therefore, alkaline leach is used at elevated temperatures of 80 to 100 degrees Celsius (176 to 212 degrees Fahrenheit) and is subjected to the hydrostatic pressure at the bottom of a 15 to 20 m (49.2 to 65.6 ft) tall tank which contains a central airlift for agitation (Figure A-18). In some mills, the leach tanks are pressurized with oxygen to increase the rate of reaction which normally takes one to three days. The alkaline leach process is characterized by the following reactions:

$2UO_2 + O_2$ --- $2UO_3$ (oxidation)

$3Na_2(CO_3) + UO_3 + H_2O$ --- $2NaOH + Na_4(UO_2)(CO_3)_3$ (leaching)

$2NaOH + CO_2$ --- $Na_2CO_3 + H_2O$ (recarbonization)

$2Na_4(UO_2)(CO_3)_3 + 6NaOH$ --- $Na_2U_2O_7 + 6Na_2CO_3 + 3H_2O$ (precipitation)

Alkaline leaching can be applied to a greater variety of ores than is currently being done; however, this process, because of its slowness, apparently involves greater capital expenditures per unit production. In addition, the purification of yellowcake, generated in a loop using sodium as the alkali element, consumes an increment of chemicals that tend to appear in stored or discharged wastewater. Purification to remove sodium ion is necessary both to meet the specifications of American uranium processors and for the preparation of natural uranium dioxide fuel. The latter process will be used to illustrate the problem caused by excess sodium. Sodium diuranate may be considered as a mixture of sodium and uranyl oxides--i.e., $Na_2U_2O_7 = Na_2O + 2UO_3$.

The process of generating UO_2 fuel pellets from a yellowcake feed involves reduction by gaseous ammonia at a temperature of a few hundred degrees C. At this temperature, ammonia thermally decomposes into hydrogen, which reduces the UO_3 component to UO_2 and nitrogen (which acts as an inert gas and reduces the risk of explosion in and around the reducing furnace). With sodium diuranate as a feed, the process results in a mix of UO_2 and Na_2O that is difficult to purify (by water leaching of NaOH) without impairing the ceramic qualities of uranium dioxide. When, in contrast, ammonium diuranate is used as the feed, all byproducts are gaseous, and pure UO_2 remains. The structural integrity of this ceramic is immediately adequate for extended use in the popular CANDU (Canadium deuterium-uranium) reactors. Sodium ion, as well as vanadium values, can be removed from raw yellowcake (sodium diuranate) produced by alkaline leaching. First, the yellowcake is roasted, and some of the sodium ion forms water-soluble sodium vanadate, while organics are carbonized and burned off. The roasted product is water leached, yielding a V_2O_5 concentrate as described below. The remaining sodium diuranate is redissolved in sulfuric acid,

$$Na_2U_2O_7 + 3H_2SO_4 \text{ --- } Na_2SO_4 + 3H_2O + 2(UO_2)SO_4$$

and the uranium values are precipitated with ammonia and filtered to yield a yellowcake (ammonium diuranate or UO_3) that is low in sodium.

$$UO_2SO_4 + H_2O + 2NH_3 \text{ --- } (NH_4)_2SO_4 + UO_3$$

The byproduct that is formed, sodium sulfate, being classed approximately in the same pollutant category as sodium chloride, requires expensive treatment for its removal. Ammonium-ion discharges, which might result from an ammonium carbonate leaching circuit, are viewed with more concern, even though there is a demand for ammonium sulfate for fertilization of alkaline southwestern soils. Ammonium sulfate could be generated by neutralizing the wastes of the ammonium loop with sulfuric acid wastes from acid leaching wastes. Opponents of a tested ammonium process argue that nitrites, an intermediate oxidation product of accidentally discharged ammonium ion, present a present health hazard more severe than from sulfate ion.

Vanadium Recovery. Vanadium, found in carnotite ($K_2(UO_2)_2(VO_4)_2 \cdot 3H_2O$) as well as in heavy metal vanadates--e.g., vanadinite ($9PbO \cdot 3V_2O_5 \cdot PbCl$)--is converted to sodium orthovanadate (Na_3VO_4), which is water-soluble, by roasting with sodium chloride or soda ash (Na_2O_3). After water leaching, ammonium chloride is added, and poorly soluble ammonium vanadates are precipitated:

$$Na_3VO_4 + 3NH_4Cl + H_2O \text{ --- } 3NaOH + NH_4VO_3 + 2NH_4OH$$

(ammonium metavanadate)

$$Na_3VO_4 + 3NH_4Cl \text{ --- } 3NaCl + (NH_4)_3VO_4$$

(ammonium orthovanadate)

The ammonium vanadates are thermally decomposed to yield vanadium pentoxide:

$$3(NH_4)_3VO_4 \text{ --- } 6NH_3 + 3H_2O + V_2O_5$$

A significant fraction (86 to 87 percent) of V_2O_5 is used in the ferroalloys industry. There, ferrovanadium has been produced in electric furnaces (the following reaction applies):

$$V_5 + Fe_2O_3 + 8C \text{ --- } 8CO + 2FeV$$

or by aluminothermic reduction (See Glossary) in the presence of scrap iron.

Air pollution problems associated with the salt roasting process have led many operators to utilize a hydrometallurgical process for vanadium recovery which is quite similar to uranium recovery by acid leaching and solvent exchange. The remainder of V_2O_5 production is used in the inorganic chemical industry.

Concentration and Precipitation. Approximately one metric ton of ore with a grade of about 0.2 percent is treated with one metric ton (or cubic meter) of leach, and the concentration(s) of uranium and/or vanadium in the pregnant solution are also about 0.2 percent. If values were directly precipitated from the solution, a significant fraction of the values would remain in solution. Therefore yellowcake is recycled and dissolved in a pregnant solution to increase precipitation yield. Direct precipitation by raising the pH is effective only for an alkaline leach, because it is more selective for uranium and vanadium. If this technique were applied to the acid leach process, most heavy metals--particularly, iron--would be precipitated, thus severely contaminating the product.

Uranium (or vanadium and molybdenum) in the pregnant leach liquor can be concentrated through ion exchange or solvent extraction. Typical concentrations in the eluate of some of the processes are shown in Table A-3.

Precipitation of uranium from the eluates is achievable without recycling yellowcake, and the selectivity of these processes under regulated conditions (particularly, pH), improves the purity of the product.

All concentration processes operate best in the absence of suspended solids, and considerable effort is made to reduce the solids content of pregnant leach liquors (Figure A-19-a). A distinction is made between quickly settling sands that are not tolerated in any concentration process and slimes that can be accomodated to some extent in the resin-in-pulp process (Figure A-19-b-c). Sands are often repulped, by the addition of some wastewater stream, to facilitate flow to the tailing pond. Consequently, there is some latitude for the selection of the wastewater sent to the tailing pond, and mill operators can take advantage of this fact in selecting environmentally sound waste disposal procedures.

Ion exchange and solvent extraction (Figure A-19-b-e) are based on the same principle: Polar organic molecules tend to exchange a mobile ion in their structure--typically, Cl^-, NO_3^-, HSO_4^-, CO_3 (anions), or H^+ or Na^+ (cations)--for an ion with a greater charge or a smaller ionic radius. For example, let R be the remainder of the polar molecule (in the case of a solvent) or polymer (for a resin), and let X be the mobile ion. Then, the exchange reaction for the uranyltrisulfate complex is:

$$4RX + (UO_2(SO_4)_3) \text{ ---- } R_4UO_2(SO_4)_3 + 4X$$

This reaction proceeds from left to right in the loading process. Typical resins adsorb about ten percent of their mass in uranium and increase by about ten percent in density. In a concentrated solution of the mobile ion--for example, in N-hydrochloric acid--the reaction can be reversed and the uranium values are eluted -- in this example, as hydrouranyl trisulfuric acid. In general, the affinity of cation exchange resins for a metallic cation increases with increasing valence (Cr^{+++}, Mg^{++}, Na^+), and because of decreasing ionic radius, with increasing atomic number (92U, 42Mo, 23V). The separation of hexavalent 92U cations by IX or SX should prove to be easier than that of any other naturally occurring element.

Uranium, vanadium, and molybdenum--the latter being a common ore consitutent--almost always appear in aqueous solutions as oxidized ions (uranyl, vanadyl, or molybdate radicals). Uranium and vanadium also combine with anionic radicals to form trisulfates or tricarbonates in the leach. The complexes react anionically, and the affinity of exchange resins and solvents is not simply related to fundamental properties of the heavy metal (uranium, vanadium, or molybdenum), as is the case in cationic exchange reactions. Secondary properties, including pH and redox potential, of the pregnant solutions influence the adsorption of heavy metals. For example, seven times more vanadium than uranium is adsorbed on one resin at pH 9 whereas at pH 11, the ratio is reversed, with 33 times as much uranium as vanadium being captured. These variations in affinity, multiple columns, and control of variations in affinity, multiple columns, and control of leaching time with respect to breakthrough (the time when the interface between loaded and regenerated resin, e.g., Figure A-19-d, arrives at the end of the column) are used to make an IX process specific for the desired product.

In the case of solvent exchange, the type of polar solvent and its concentration in a typically nonpolar diluent (e.g., kerosene) effect separation of the desired product. The ease with which the solvent is handled (Figure A-19-e) permits the construction of multistage co-current and countercurrent SX concentrators that are useful even when each stage effects only partial separation of a value from an interferent. Unfortunately, the solvents are easily polluted by slimes, and complete liquid/solid separation is necessary. IX and SX circuits can be combined to take advantage of both the slime resistance of resin-in-pulp ion exchange and the separatory efficiency of solvent exchange (Eluex process-Figure A-19-f). The uranium values are precipitated with a base or a combination of base and hydrogen peroxide. Ammonia is preferred by a plurality of mills because it results in a superior product, as mentioned in the discussion of alkaline leaching. Sodium hydroxide, magnesium hydroxide, or partial neutralization with calcium hydroxide followed by magnesium hydroxide precipitation, are also used. The product is rinsed with water that is recycled into the process to preserve values, then filtered, dried and packed into 200-liter (55-gallon) drums. The strength of these drums limits their capacity to 450 kg (1000 pounds) of yellowcake which occupies 28 percent of the drum volume.

Figure A-19-g illustrates the Split Elution Concentration process. Figure A-20 illustrates a "Generalized Flow Diagram for Production of Uranium, Vanadium and Radium."

Antimony Ore Milling Processes

Antimony is recovered from antimony ore and as a byproduct from silver and lead concentrates.

Only a small percentage of antimony (13 percent in 1972) is recovered from ore being mined primarily for its antimony content. Nearly all of this production can be attributed to a single operation which is using a froth flotation process to concentrate stibnite (Sb_2S_3) (Figure A-21).

The bulk of domestic production of antimony is recovered as a byproduct of silver mining operations in the Coeur d'Alene district of Idaho. Antimony is present in the silver-containing mineral tetrahedrite and is recovered from tetrahedrite concentrates in an electrolytic antimony extraction plant owned and operated by one of the silver mining companies in the Coeur d'Alene district. Mills are usually penalized for the antimony content in their concentrates. Therefore, the removal of antimony from the tetrahedrite concentrates not only increases their value, but the antimony itself then becomes a marketableiitem.

Antimony is also contained in lead concentrates and is ultimately recovered as a byproduct at lead smelters--usually as antimonial lead. This source of antimony represents about 30 to 50 percent of domestic production in recent years.

Titanium Ore Milling Processes

The method of mining and beneficiating titanium minerals depends upon whether the ore is contained ina sand or rock deposit. Sand deposits occurring in Florida, Georgia, and New Jerrsey contain 1 to 5 percent TiO_2% and are mined with floating suction or bucket-line dredges handling up to 1,088 metric tons (1,200 short tons) of material per hour. The sand is treated by wet graiity methods using spirals, cones, sluices, or jigs to produce a bulk, mixed, heavy-mineral concentra As many as five individual marketable minerals are then separated from the bulk concentrate by a cbination of dry separation techniques using magnetic and electrostatic (high-tension) separators, sometimes in conjunction with dry and wet gravity concentrating equipment.

High-tension (HT) electrostatic separators are employed to separate the titanium minerals from the silicate minerals. The minerals are fed onto a high-speed spinning rotor, and a heavy corona (glow given off by a high voltage charge) discharge is aimed toward the minerals at the point where they would normally leave the rotor. The minerals of relatively poor electrical conductance are pinned to the rotor by the high surface charge they recieve on passing through the high voltage corona. The minerals of relatively high conductivity do not readily hold this surface charge and so leave the rotor in their normal trajectory. Titanium minerals are the only ones present of relatively high electrical conductivity and are, therefore, thrown off the rotor. The silicates are pinned to the rotor and are removed by a fixed brush.

Titanium minerals undergo final separation in induced-roll magnetic separators to produce three products: ilmenite, leucoxine, and rutile. The separation of these minerals is based on their relative magnetic properties which, in turn, are based on their relative iron content: ilmenite has 37 to 65 percent iron, leucoxine has 30 to 40 percent iron, and rutile has 4 to 10 percent iron.

Tailings from the HT separators (nonconductors) may contain zircon and monazite (a rare-earth mineral). These heavy minerals are separated from the other nonconductors (silicates) by various wet gravity methods (i.e., spirals or tables). The zircon (non-magnetic) and monazite (slightly magnetic) are separated from one another in induced-roll magnetic separators.

Beneficiation of titanium minerals from beach-sand deposits is illustrated in Figure A-22.

Ilmenite is also currently mined from a rock deposit in New York by conventional open-pit methods. This ilmenite/magnetite ore, averaging 18 percent TiO_2, is crushed and ground to a small particle size. The ilmenite and magnetite fractions are separated in a magnetic separator, the magnetite being more magnetic due to its greater iron content. The ilmenite sands are further upgraded in a flotation circuit. Beneficiation of titanium from a rock deposit is illustrated in Figure A-23.

Table A-1

REAGENTS USED FOR FLOTATION OF IRON ORES

(Reagent quantities represent approximate maximum usages. Exact chemical composition of reagent may be unknown.)

1. **Anionic Flotation of Iron Oxides** (from crude ore)

   Petroleum sulfonate: 0.5 kg/metric ton (1 lb/short ton)
   Low-rosin, tall oil fatty acid: 0.25 kg/metric ton (0.5 lb/short ton)
   Sulfuric acid: 1.25 kg/metric ton (2.5 lb/short ton) to pH3
   No. 2 fuel oil: 0.15 kg/metric ton (0.3 lb/short ton)
   Sodium silicate: 0.5 kg/metric ton (1 lb/short ton)

2. **Anionic Flotation of Iron Oxides** (from crude ore)

   Low-rosin tall oil fatty acid: 0.5 kg/metric ton (1 lb/short ton)

3. **Cationic Flotation of Hematite** (from crude ore)

   Rosin amine acetate: 0.2 kg/metric ton (0.4 lb/short ton)
   Sulfuric acid: 0.15 kg/metric ton (0.3 lb/short ton)
   Sodium fluoride: 0.15 kg/metric ton (0.3 lb/short ton)
   (Plant also includes phosphate flotation and pyrite flotation steps. Phosphate flotation employs sodium hydroxide, tall oil fatty acid, fuel oil, and sodium silicate. Pyrite flotation employs xanthate collector.)

4. **Cationic Flotation of Silica** (from crude ore)

   Amine: 0.15 kg/metric ton (0.3 lb/short ton)
   Gum or starch (tapioca fluor): 0.5 kg/metric ton (1 lb/short ton)
   Methylisobutyl carbinol: as required

5. **Cationic Flotation of Silica** (from magnetite concentrate)

   Amine: 5 g/metric ton (0.01 lb/short ton)
   Methylisobutyl carbinol: as required

Table A-2

VARIOUS FLOTATION METHODS AVAILABLE FOR PRODUCTION OF HIGH-GRADE IRON-ORE CONCENTRATE

1. **Anionic flotation of specular hematite**
2. **Upgrading of natural magnetite concentrate by cationic flotation**
3. **Upgrading of artificial magnetite concentrate by cationic flotation**
4. **Cationic flotation of crude magnetite**
5. **Anionic flotation of silica from natural hematite**
6. **Cationic flotation of silica from non-magnetic iron formation**

Table A-3

URANIUM CONCENTRATION IN IX/SX ELUATES

PROCESS	U_3O_8 CONCENTRATION (%)
Ion exchange	
Resin-in-pulp	0.8 to 1.2
Fixed-bed IX:	
Chloride elution	0.5 to 1.0
Nitrate elution	1.0 to 2.0
Moving-bed IX:	
Nitrate elution	1.9
Solvent extraction	
Alkyl phosphates, HCl eluent	30.0 to 60.0
Amex process	3 to 4
Dapex process	5.0 to 6.5
Split elution minewater treatment	1.2 to 1.6
IX/SX combination	
Eluex process	3.0 to 7.5

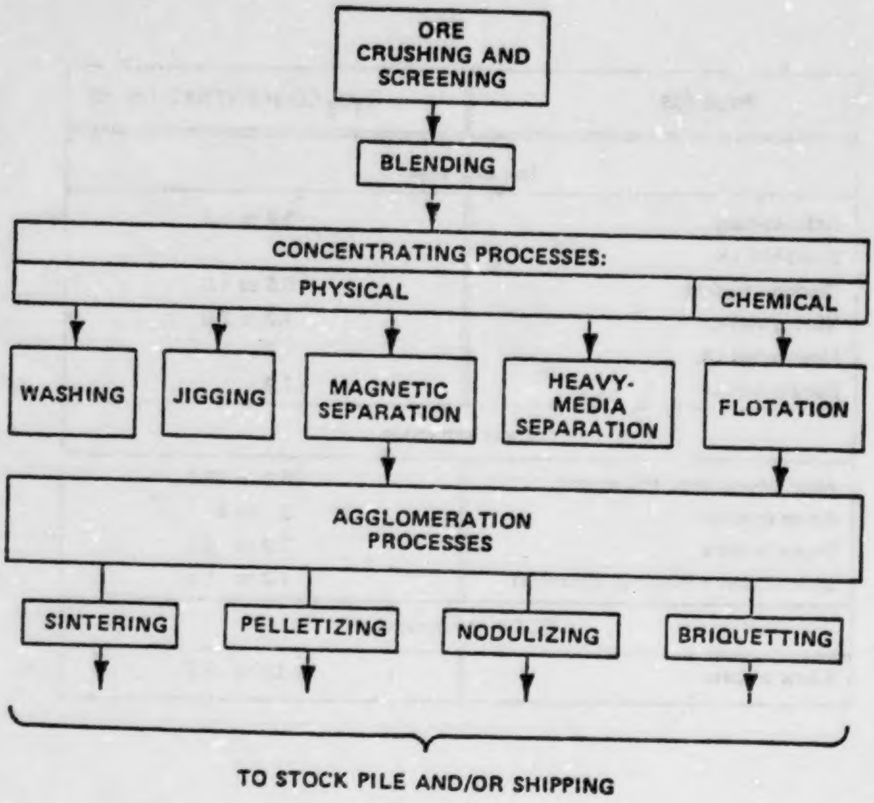

ORE CRUSHING AND SCREENING
BLENDING
CONCENTRATING PROCESSES:
PHYSICAL
CHEMICAL
WASHING
JIGGING
MAGNETIC SEPARATION
HEAVY-MEDIA SEPARATION
FLOTATION
AGGLOMERATION PROCESSES
SINTERING
PELLETIZING
NODULIZING
BRIQUETTING
TO STOCK PILE AND/OR SHIPPING

Figure A-1

BENEFICIATION OF IRON ORES

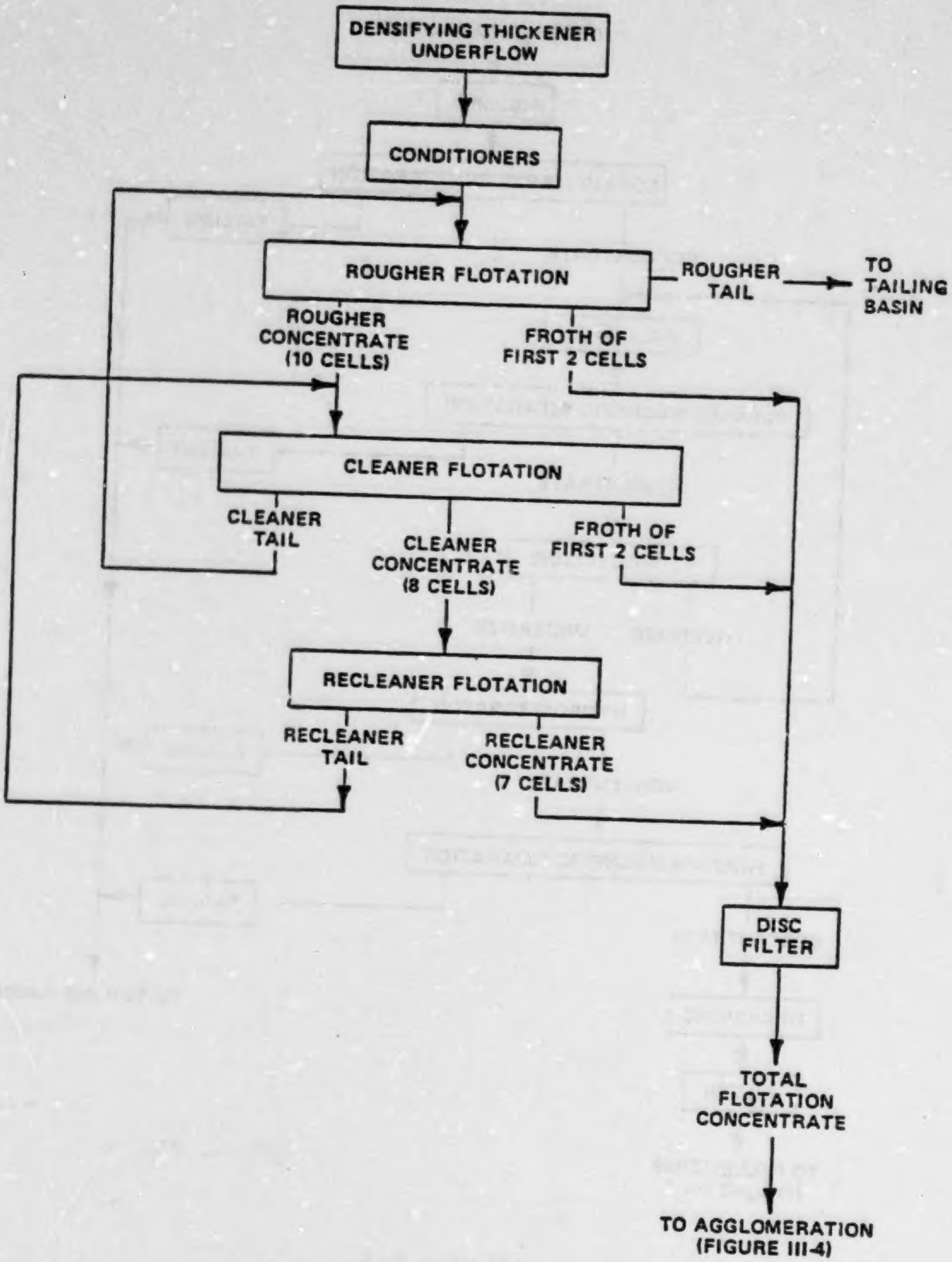

DENSIFYING THICKENER UNDERFLOW
CONDITIONERS
ROUGHER FLOTATION
ROUGHER TAIL
TO TAILING BASIN
ROUGHER CONCENTRATE (10 CELLS)
FROTH OF FIRST 2 CELLS
CLEANER FLOTATION
CLEANER TAIL
CLEANER CONCENTRATE (8 CELLS)
FROTH OF FIRST 2 CELLS
RECLEANER FLOTATION
RECLEANER TAIL
RECLEANER CONCENTRATE (7 CELLS)
DISC FILTER
TOTAL FLOTATION CONCENTRATE
TO AGGLOMERATION (FIGURE III-4)

Figure A-2
IRON-ORE FLOTATION-CIRCUIT FLOWSHEET

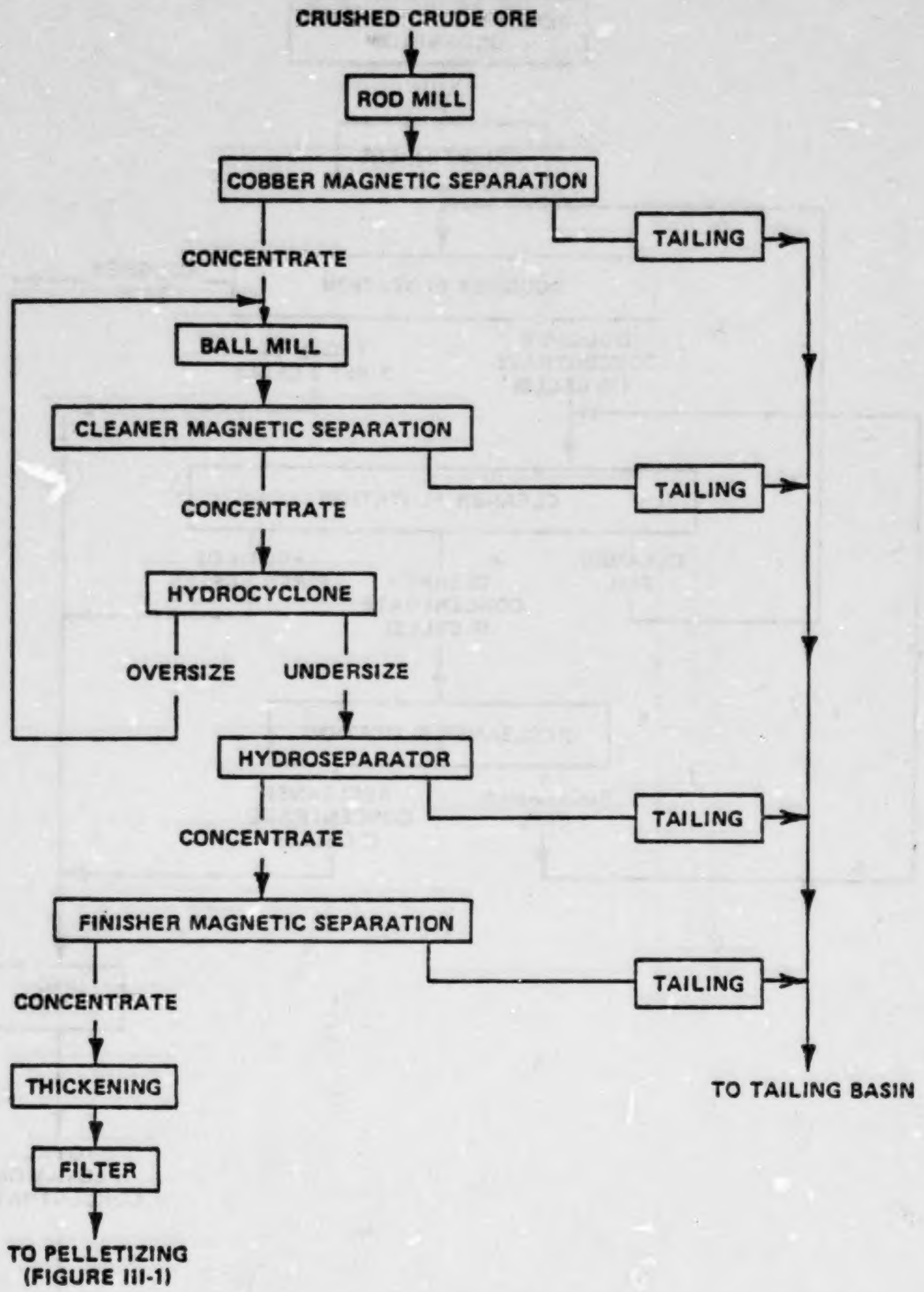

CRUSHED CRUDE ORE
ROD MILL
COBBER MAGNETIC SEPARATION
TAILING
CONCENTRATE
BALL MILL
CLEANER MAGNETIC SEPARATION
TAILING
CONCENTRATE
HYDROCYCLONE
OVERSIZE
UNDERSIZE
HYDROSEPARATOR
TAILING
CONCENTRATE
FINISHER MAGNETIC SEPARATION
TAILING
CONCENTRATE
THICKENING
FILTER
TO PELLETIZING
(FIGURE III-1)
TO TAILING BASIN

Figure A-3

MAGNETIC TACONITE BENEFICIATION FLOWSHEET

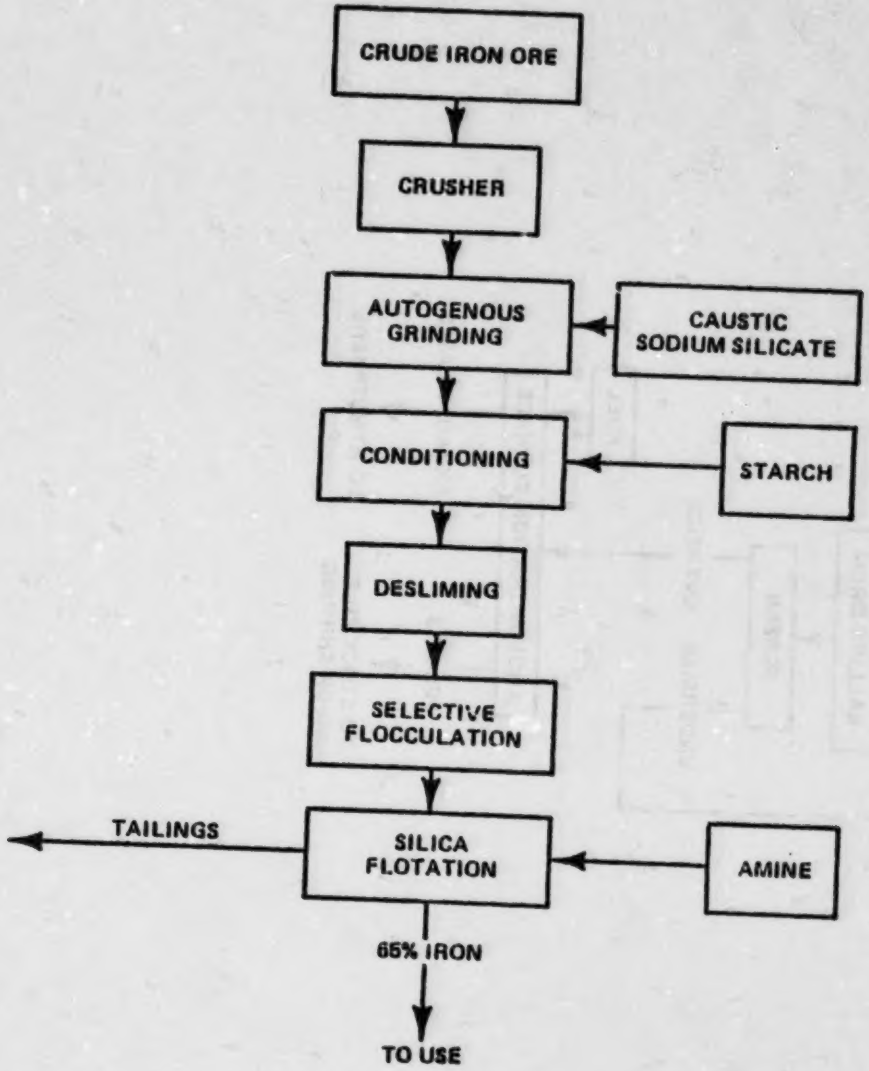

CRUDE IRON ORE
CRUSHER
AUTOGENOUS GRINDING
CAUSTIC SODIUM SILICATE
CONDITIONING
STARCH
DESLIMING
SELECTIVE FLOCCULATION
TAILINGS
SILICA FLOTATION
AMINE
65% IRON
TO USE

Figure A-4

SIMPLIFIED FLOWSHEET FOR FINE-GRAINED HEMATITE BENEFICIATION

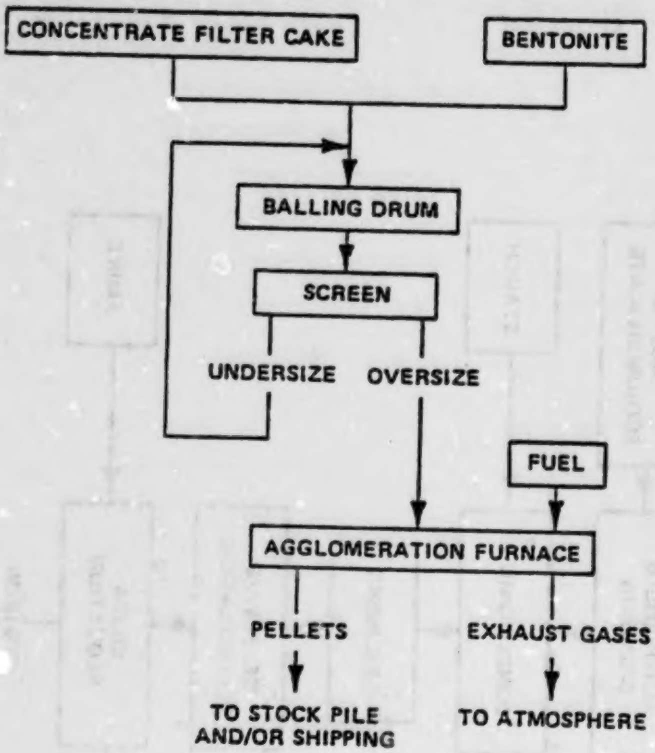

CONCENTRATE FILTER CAKE
BENTONITE
BALLING DRUM
SCREEN
UNDERSIZE
OVERSIZE
FUEL
AGGLOMERATION FURNACE
PELLETS
EXHAUST GASES
TO STOCK PILE AND/OR SHIPPING
TO ATMOSPHERE

Figure A-5

AGGLOMERATION FLOWSHEET

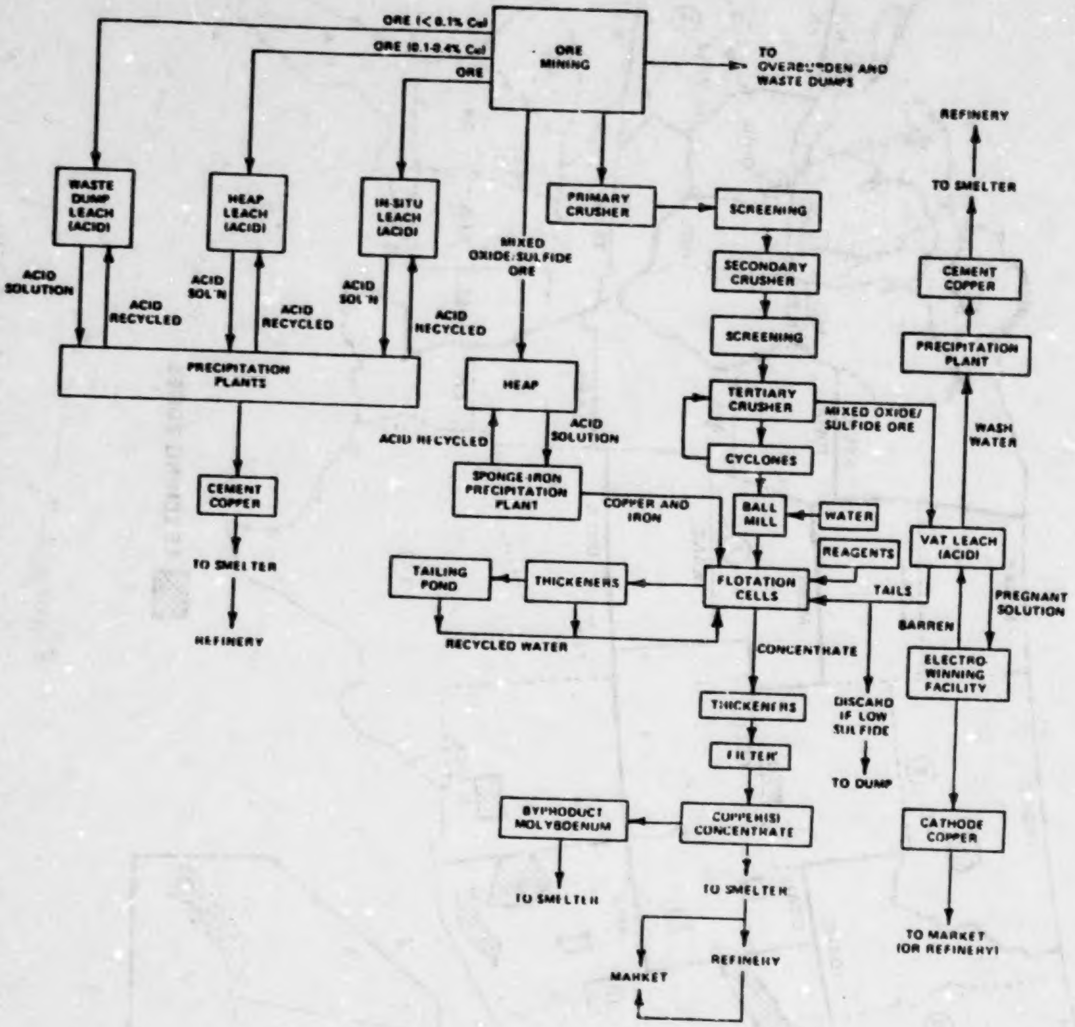

ORE (< 0.1% Cu)
ORE (0.1-0.4% Cu)
ORE
ORE MINING
TO OVERBURDEN AND WASTE DUMPS
WASTE DUMP LEACH (ACID)
HEAP LEACH (ACID)
IN-SITU LEACH (ACID)
ACID SOLUTION
ACID RECYCLED
ACID SOL'N
ACID RECYCLED
ACID SOL'N
ACID RECYCLED
PRECIPITATION PLANTS
CEMENT COPPER
TO SMELTER
REFINERY
MIXED OXIDE/SULFIDE ORE
HEAP
ACID RECYCLED
ACID SOLUTION
SPONGE-IRON PRECIPITATION PLANT
COPPER AND IRON
PRIMARY CRUSHER
SCREENING
SECONDARY CRUSHER
SCREENING
TERTIARY CRUSHER
MIXED OXIDE/ SULFIDE ORE
CYCLONES
BALL MILL
WATER
REAGENTS
TAILING POND
THICKENERS
FLOTATION CELLS
RECYCLED WATER
TAILS
CONCENTRATE
THICKENERS
FILTER
DISCARD IF LOW SULFIDE
TO DUMP
BYPRODUCT MOLYBDENUM
TO SMELTER
COPPER(S) CONCENTRATE
TO SMELTER
REFINERY
MARKET
REFINERY
TO SMELTER
CEMENT COPPER
PRECIPITATION PLANT
WASH WATER
VAT LEACH (ACID)
BARREN
PREGNANT SOLUTION
ELECTRO-WINNING FACILITY
CATHODE COPPER
TO MARKET (OR REFINERY)

Figure A-6

GENERAL FLOW DIAGRAM DEPICTING METHODS FOR TYPICAL RECOVERY OF COPPER FROM ORE

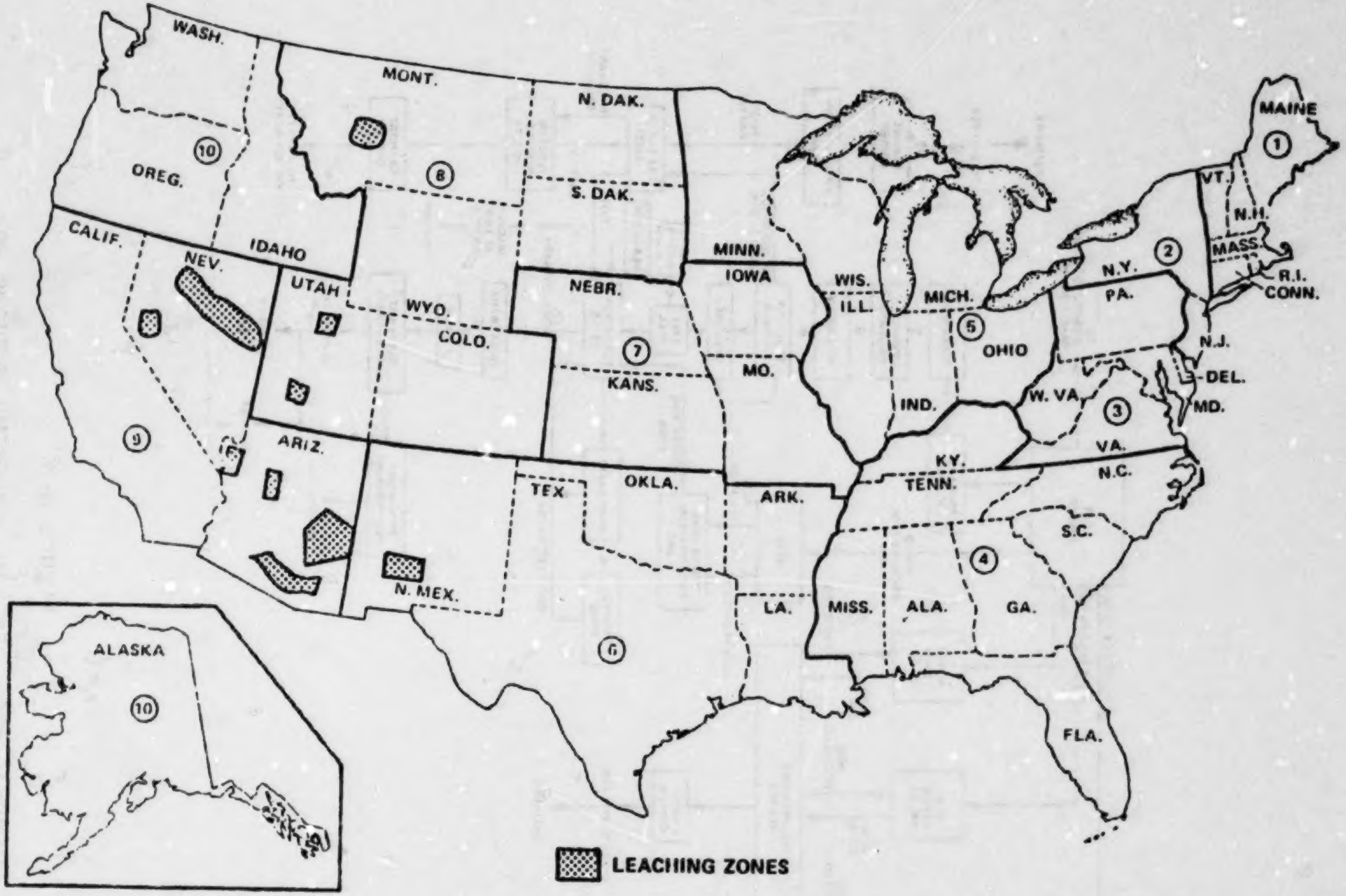

WASH.
MONT.
N. DAK.
MAINE
①
⑩
OREG.
⑧
S. DAK.
VT.
N.H.
CALIF.
IDAHO
MINN.
MASS.
②
NEV.
UTAH
NEBR.
IOWA
WIS.
N.Y.
R.I.
CONN.
ILL.
MICH.
PA.
WYO.
COLO.
⑦
⑤
OHIO
N.J.
KANS.
MO.
DEL.
IND.
W. VA.
③
MD.
⑨
ARIZ.
VA.
KY.
TEX.
OKLA.
N.C.
TENN.
ARK.
S.C.
④
N. MEX.
LA.
MISS.
ALA.
GA.
ALASKA
⑥
⑩
FLA.
▨ LEACHING ZONES

Figure A-7

MAJOR COPPER AREAS EMPLOYING ACID LEACHING IN DUMPS, HEAPS OR IN-SITU

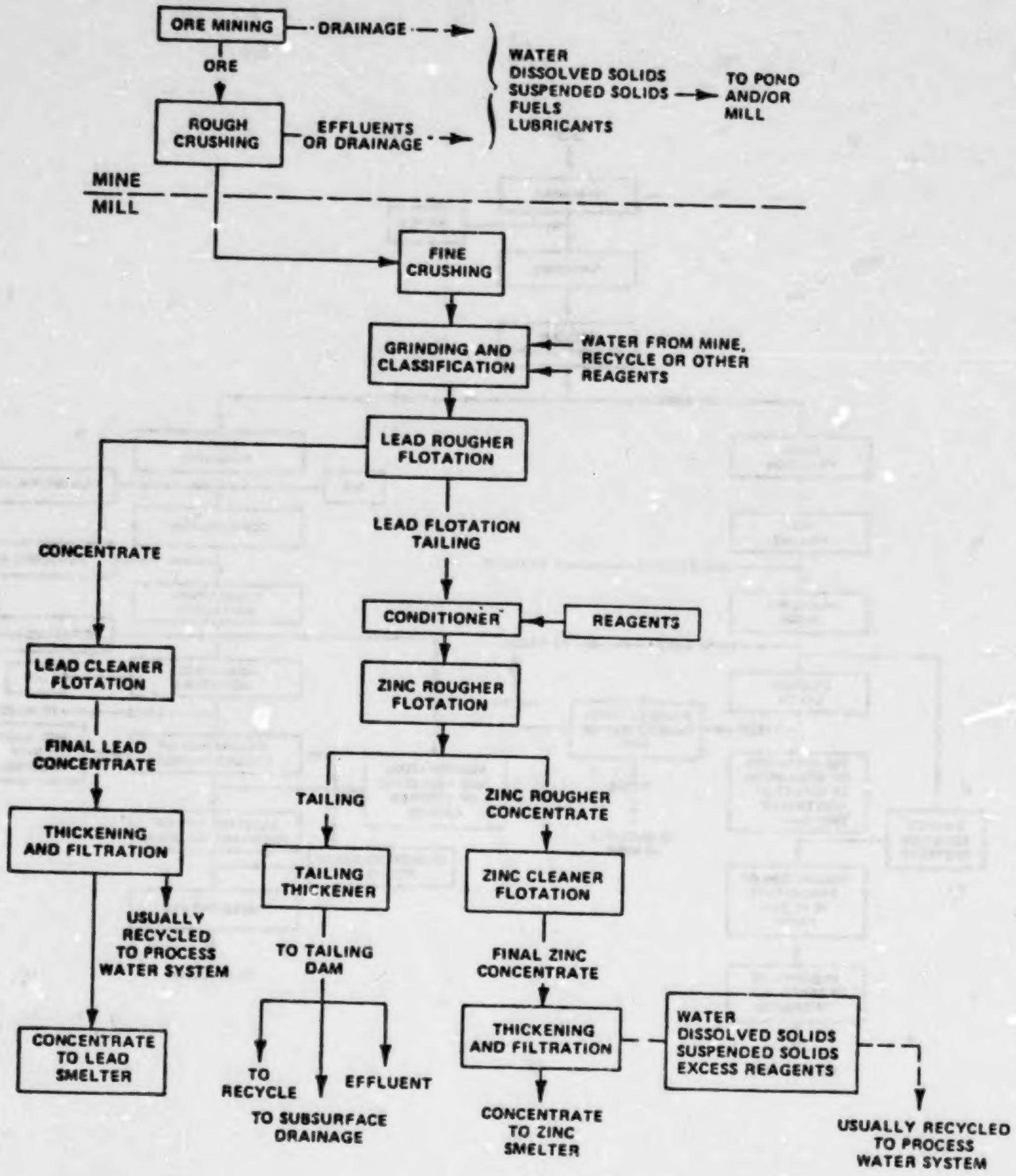

ORE MINING
DRAINAGE
ORE
ROUGH CRUSHING
EFFLUENTS OR DRAINAGE
WATER
DISSOLVED SOLIDS
SUSPENDED SOLIDS
FUELS
LUBRICANTS
TO POND AND/OR MILL
MINE
MILL
FINE CRUSHING
GRINDING AND CLASSIFICATION
WATER FROM MINE, RECYCLE OR OTHER REAGENTS
LEAD ROUGHER FLOTATION
CONCENTRATE
LEAD FLOTATION TAILING
CONDITIONER
REAGENTS
LEAD CLEANER FLOTATION
ZINC ROUGHER FLOTATION
FINAL LEAD CONCENTRATE
TAILING
ZINC ROUGHER CONCENTRATE
THICKENING AND FILTRATION
TAILING THICKENER
ZINC CLEANER FLOTATION
USUALLY RECYCLED TO PROCESS WATER SYSTEM
TO TAILING DAM
FINAL ZINC CONCENTRATE
CONCENTRATE TO LEAD SMELTER
TO RECYCLE
TO SUBSURFACE DRAINAGE
EFFLUENT
THICKENING AND FILTRATION
WATER
DISSOLVED SOLIDS
SUSPENDED SOLIDS
EXCESS REAGENTS
CONCENTRATE TO ZINC SMELTER
USUALLY RECYCLED TO PROCESS WATER SYSTEM

Figure A-8

TYPICAL LEAD-ZINC MINING AND PROCESSING OPERATION

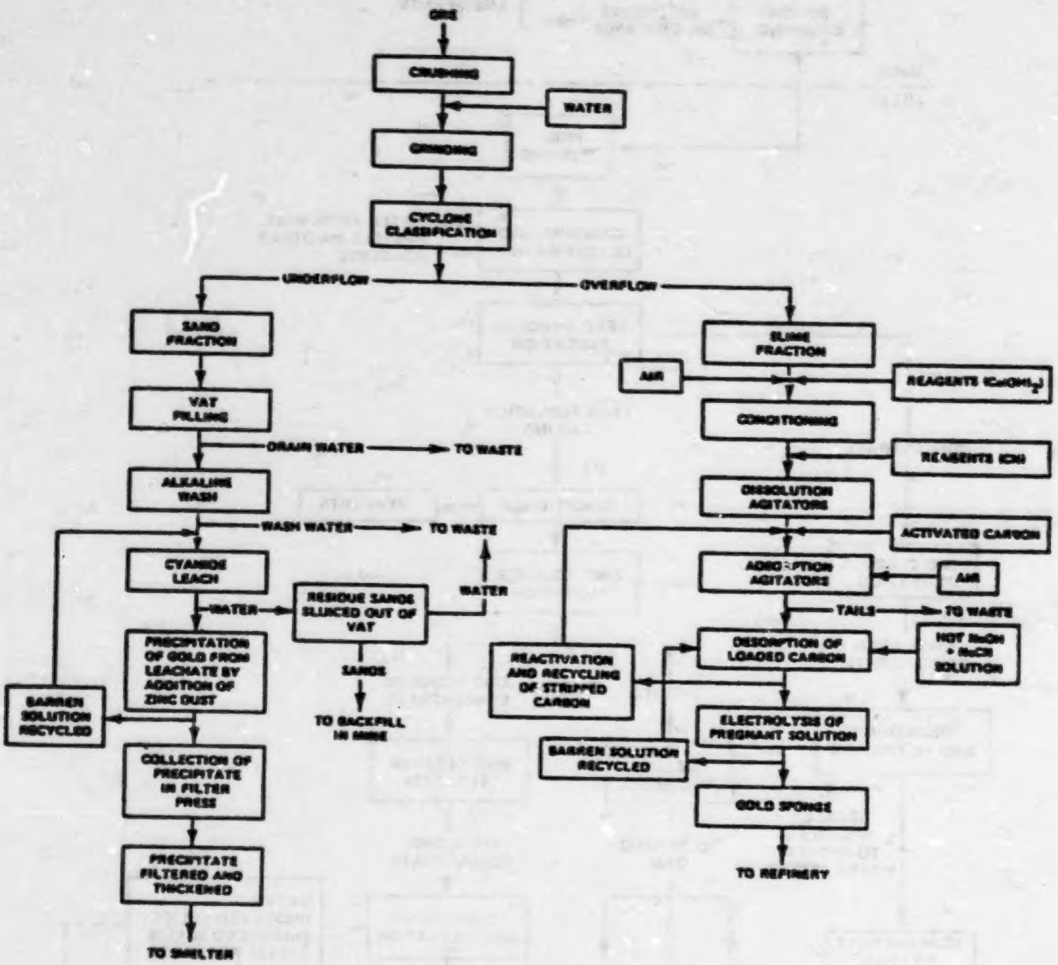

ORE
CRUSHING
WATER
GRINDING
CYCLONE CLASSIFICATION
UNDERFLOW
OVERFLOW
SAND FRACTION
VAT FILLING
DRAIN WATER
TO WASTE
ALKALINE WASH
WASH WATER
TO WASTE
CYANIDE LEACH
WATER
RESIDUE SANDS SLUICED OUT OF VAT
WATER
SANDS
TO BACKFILL IN MINE
PRECIPITATION OF GOLD FROM LEACHATE BY ADDITION OF ZINC DUST
BARREN SOLUTION RECYCLED
COLLECTION OF PRECIPITATE IN FILTER PRESS
PRECIPITATE FILTERED AND THICKENED
TO SMELTER
SLIME FRACTION
AIR
REAGENTS (Ca(OH)2)
CONDITIONING
REAGENTS (CN)
DISSOLUTION AGITATORS
ACTIVATED CARBON
ADSORPTION AGITATORS
AIR
TAILS
TO WASTE
DESORPTION OF LOADED CARBON
HOT NaOH + NaCN SOLUTION
REACTIVATION AND RECYCLING OF STRIPPED CARBON
ELECTROLYSIS OF PREGNANT SOLUTION
BARREN SOLUTION RECYCLED
GOLD SPONGE
TO REFINERY

Figure A-9

CYANIDATION OF GOLD ORE: VAT LEACHING OF SANDS AND 'CARBON-IN-PULP' PROCESSING OF SLIMES

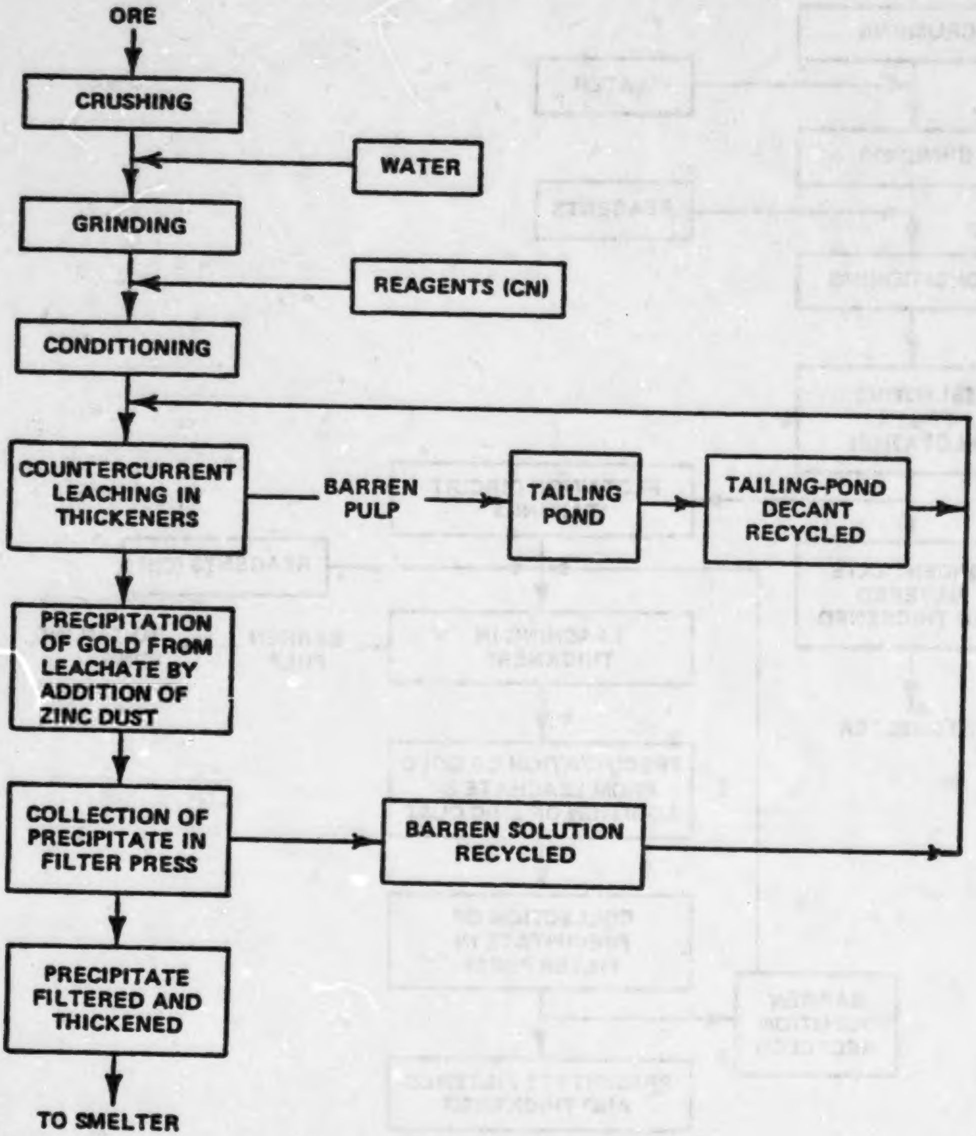

ORE
CRUSHING
WATER
GRINDING
REAGENTS (CN)
CONDITIONING
COUNTERCURRENT LEACHING IN THICKENERS
BARREN PULP
TAILING POND
TAILING-POND DECANT RECYCLED
PRECIPITATION OF GOLD FROM LEACHATE BY ADDITION OF ZINC DUST
COLLECTION OF PRECIPITATE IN FILTER PRESS
BARREN SOLUTION RECYCLED
PRECIPITATE FILTERED AND THICKENED
TO SMELTER

Figure A-10

CYANIDATION OF GOLD ORE: AGITATION/LEACH PROCESS

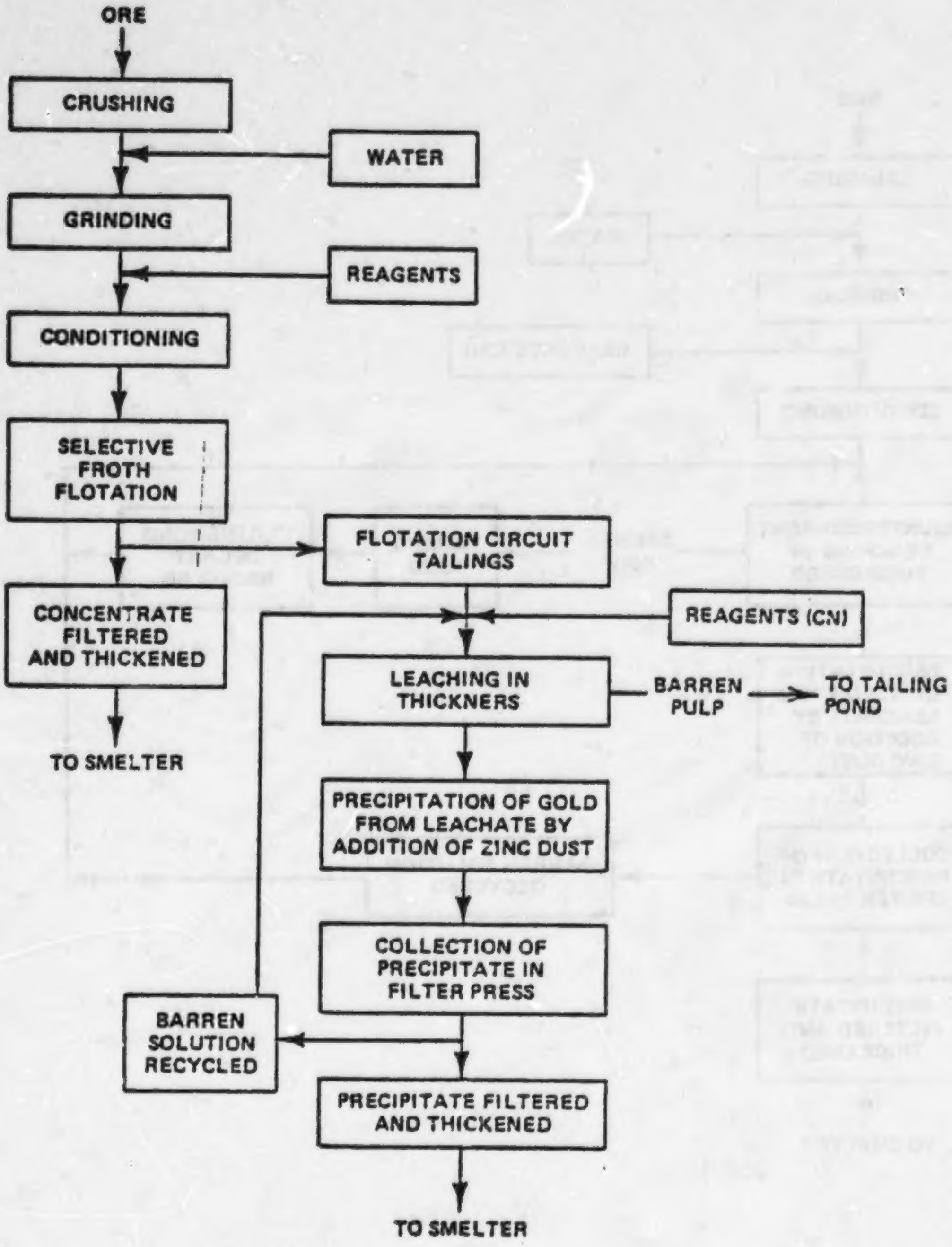
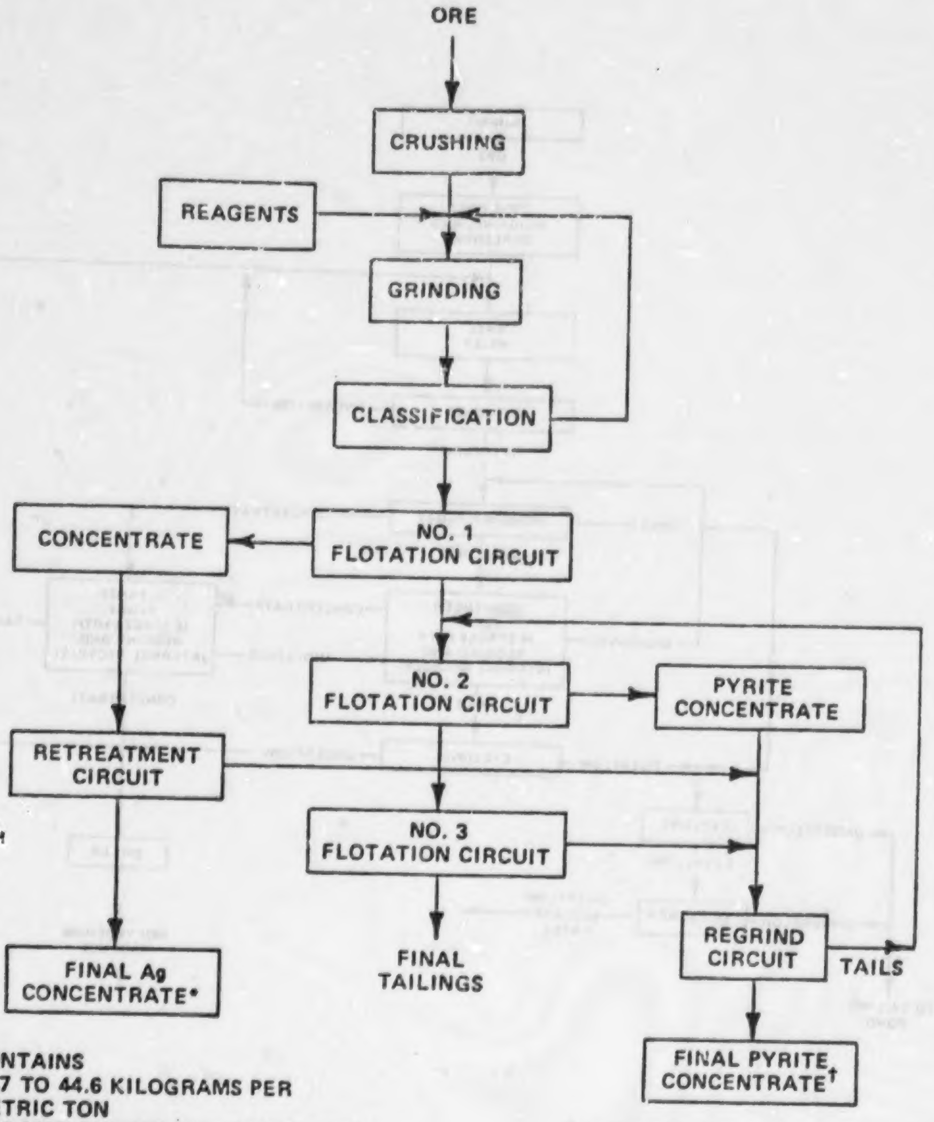

ORE
CRUSHING
WATER
GRINDING
REAGENTS
CONDITIONING
SELECTIVE FROTH FLOTATION
FLOTATION CIRCUIT TAILINGS
CONCENTRATE FILTERED AND THICKENED
REAGENTS (CN)
LEACHING IN THICKNERS
BARREN PULP
TO TAILING POND
TO SMELTER
PRECIPITATION OF GOLD FROM LEACHATE BY ADDITION OF ZINC DUST
COLLECTION OF PRECIPITATE IN FILTER PRESS
BARREN SOLUTION RECYCLED
PRECIPITATE FILTERED AND THICKENED
TO SMELTER

Figure A-11

FLOTATION OF GOLD-CONTAINING MINERALS WITH RECOVERY OF RESIDUAL GOLD VALUES BY CYANIDATION


ORE
CRUSHING
REAGENTS
GRINDING
CLASSIFICATION
CONCENTRATE
NO. 1 FLOTATION CIRCUIT
NO. 2 FLOTATION CIRCUIT
PYRITE CONCENTRATE
RETREATMENT CIRCUIT
NO. 3 FLOTATION CIRCUIT
FINAL Ag CONCENTRATE*
FINAL TAILINGS
REGRIND CIRCUIT
TAILS
FINAL PYRITE CONCENTRATE†

*CONTAINS
25.7 TO 44.6 KILOGRAMS PER METRIC TON
(750-1300 OUNCES PER SHORT TON):
25 TO 32% COPPER
0 TO 18% ANTIMONY

†CONTAINS 3.43 KILOGRAMS PER METRIC TON (100 TROY OUNCES PER SHORT TON)

Figure A-12

RECOVERY OF SILVER ORE BY FROTH FLOTATION

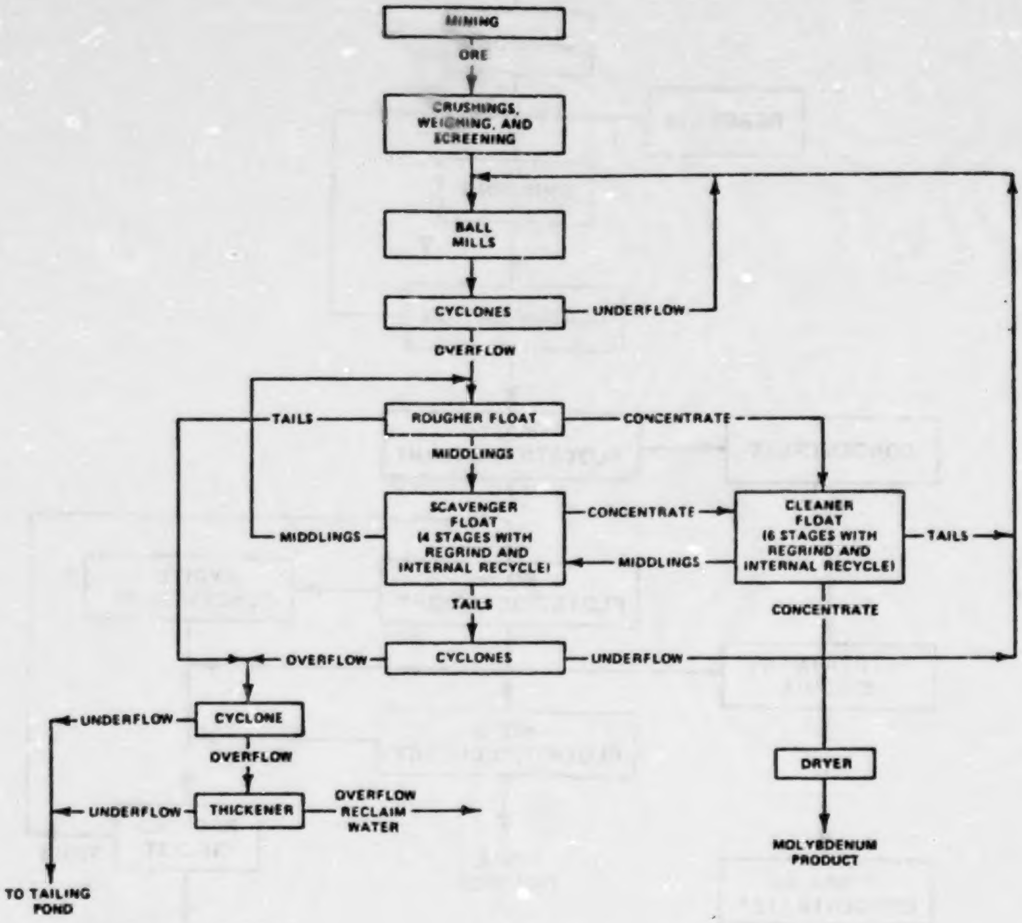

MINING
ORE
CRUSHINGS, WEIGHING, AND SCREENING
BALL MILLS
CYCLONES
UNDERFLOW
OVERFLOW
TAILS
ROUGHER FLOAT
CONCENTRATE
MIDDLINGS
MIDDLINGS
SCAVENGER FLOAT (4 STAGES WITH REGRIND AND INTERNAL RECYCLE)
CONCENTRATE
CLEANER FLOAT (6 STAGES WITH REGRIND AND INTERNAL RECYCLE)
TAILS
MIDDLINGS
TAILS
CONCENTRATE
OVERFLOW
CYCLONES
UNDERFLOW
UNDERFLOW
CYCLONE
OVERFLOW
UNDERFLOW
THICKENER
OVERFLOW RECLAIM WATER
DRYER
MOLYBDENUM PRODUCT
TO TAILING POND

Figure A-13

SIMPLIFIED MOLYBDENUM MILL FLOWSHEET
SHOWING RECOVERY OF MOLYBDENITE ONLY

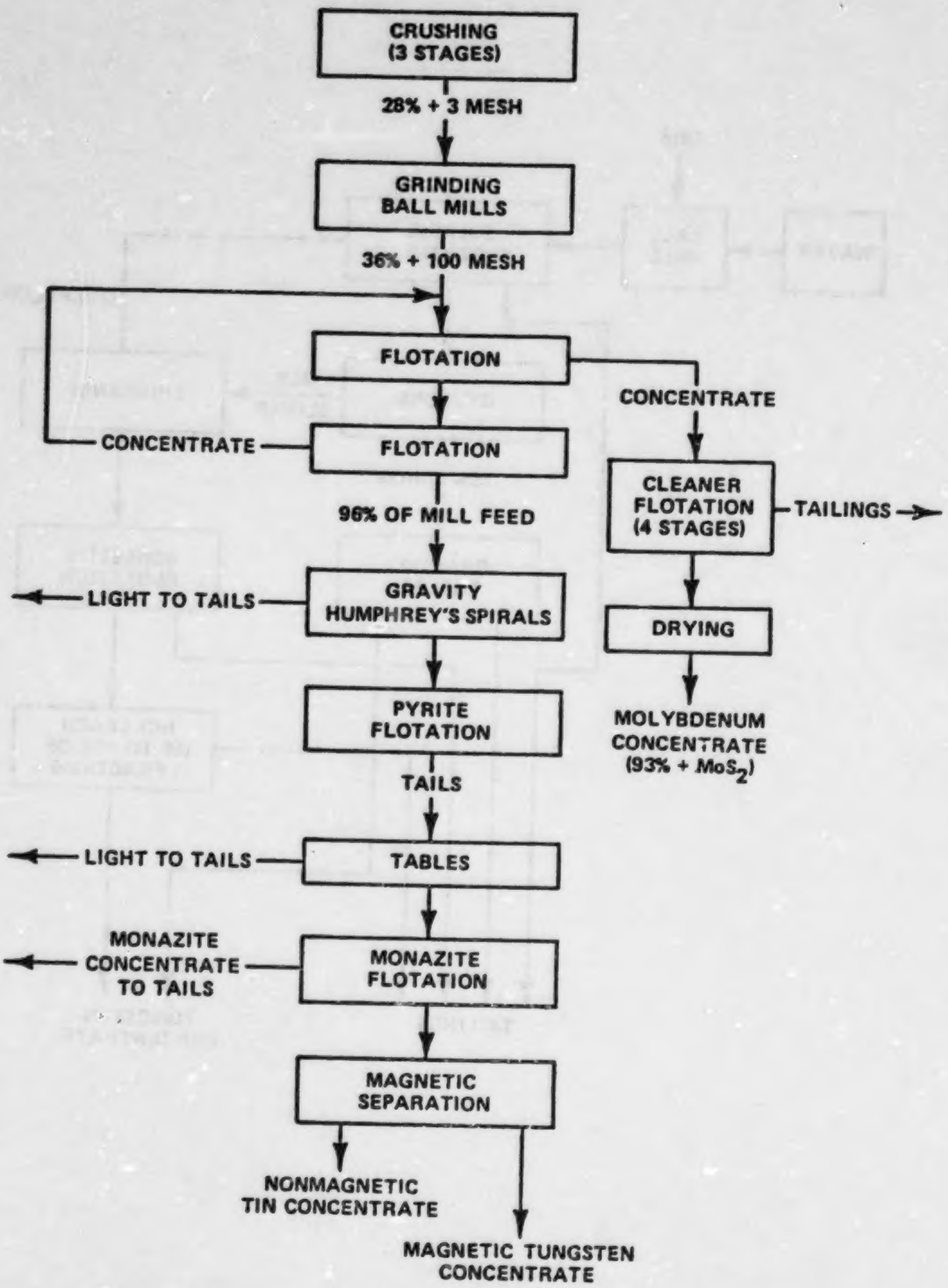

CRUSHING (3 STAGES)
28% + 3 MESH
GRINDING BALL MILLS
36% + 100 MESH
FLOTATION
CONCENTRATE
CONCENTRATE
FLOTATION
CLEANER FLOTATION (4 STAGES)
TAILINGS
96% OF MILL FEED
LIGHT TO TAILS
GRAVITY HUMPHREY'S SPIRALS
DRYING
PYRITE FLOTATION
MOLYBDENUM CONCENTRATE (93% + MoS_2)
TAILS
LIGHT TO TAILS
TABLES
MONAZITE CONCENTRATE TO TAILS
MONAZITE FLOTATION
MAGNETIC SEPARATION
NONMAGNETIC TIN CONCENTRATE
MAGNETIC TUNGSTEN CONCENTRATE

Figure A-14

SIMPLIFIED MOLYBDENUM MILL FLOW DIAGRAM
SHOWING RECOVERY OF MOLYBDENITE AND BYPRODUCTS

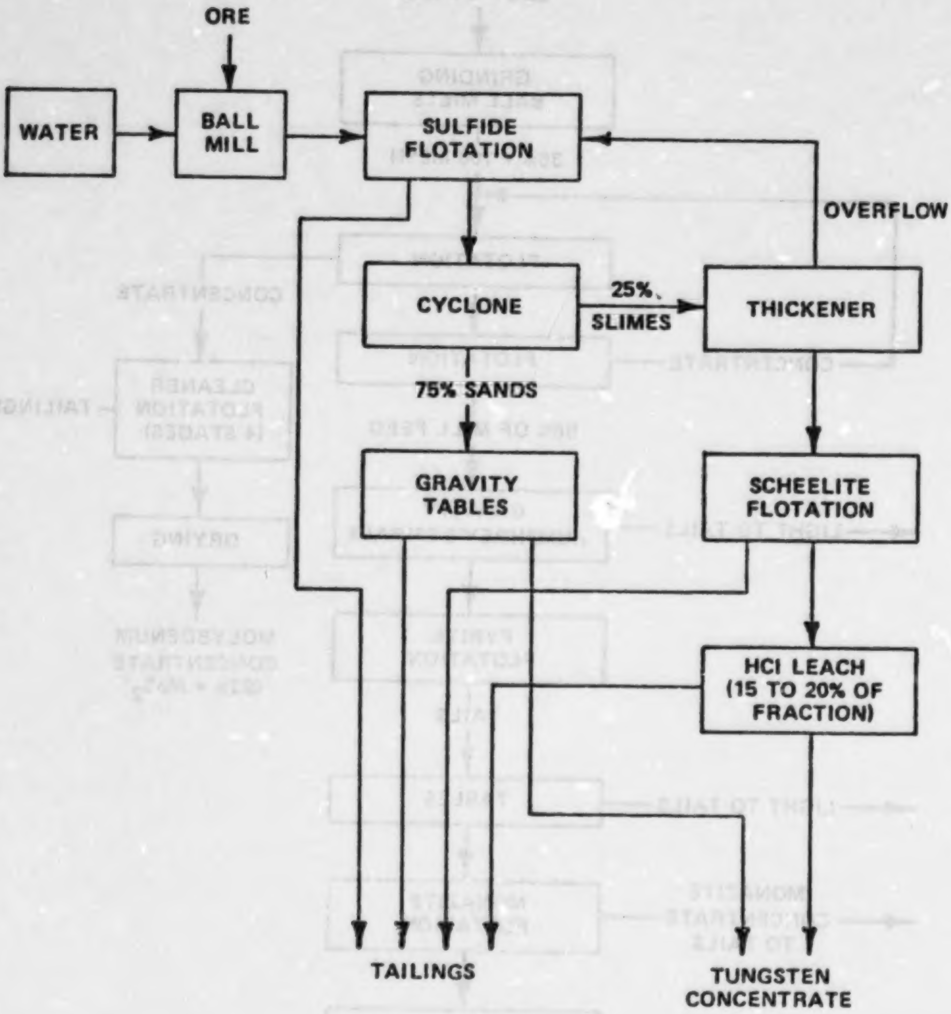

ORE
WATER
BALL MILL
SULFIDE FLOTATION
OVERFLOW
CYCLONE
25% SLIMES
THICKENER
75% SANDS
GRAVITY TABLES
SCHEELITE FLOTATION
HCl LEACH (15 TO 20% OF FRACTION)
TAILINGS
TUNGSTEN CONCENTRATE

Figure A-15

SIMPLIFIED FLOW DIAGRAM FOR A SMALL TUNGSTEN CONCENTRATOR

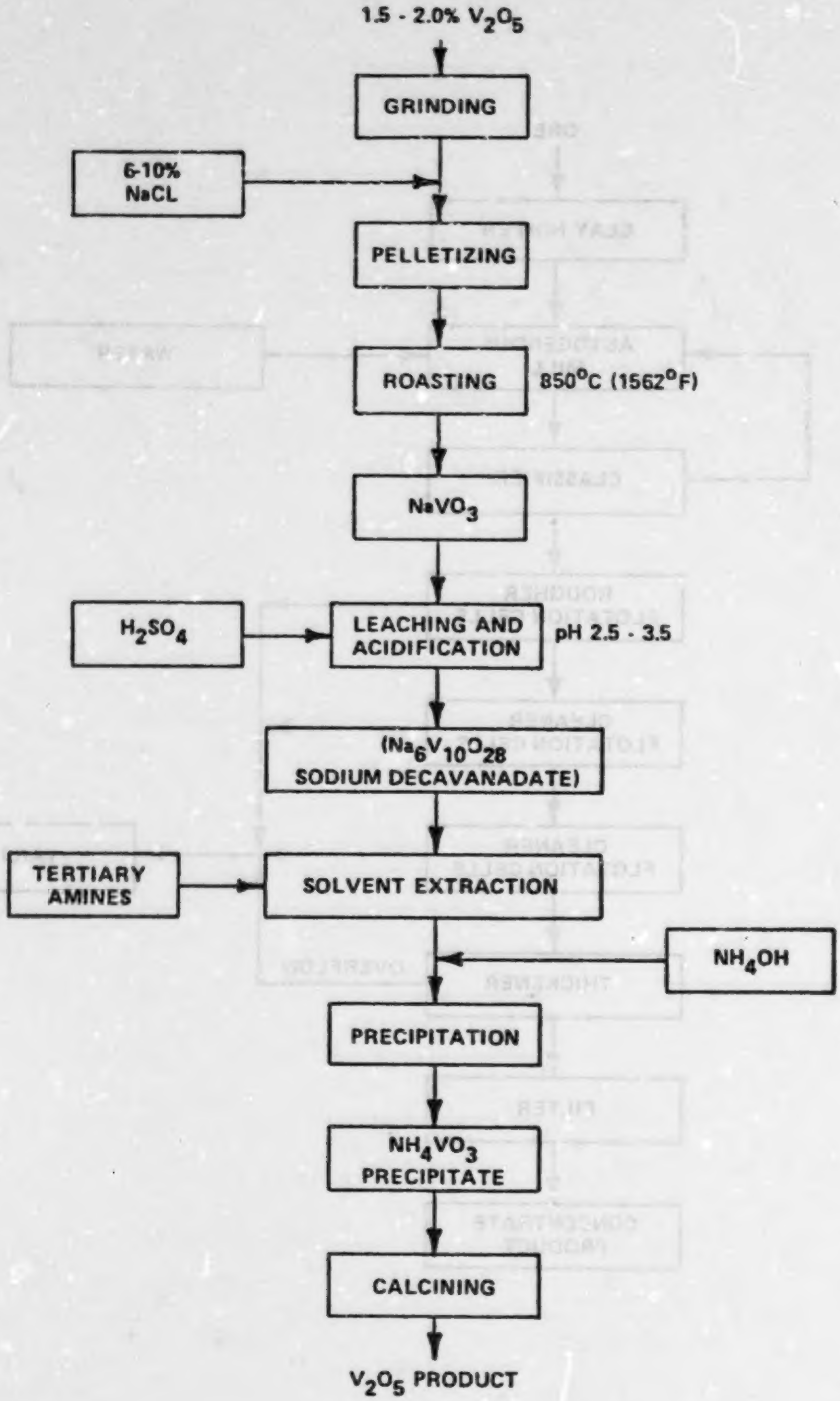

1.5 - 2.0% V2O5
GRINDING
6-10% NaCL
PELLETIZING
ROASTING
850°C (1562°F)
NaVO3
H2SO4
LEACHING AND ACIDIFICATION
pH 2.5 - 3.5
(Na6V10O28 SODIUM DECAVANADATE)
TERTIARY AMINES
SOLVENT EXTRACTION
NH4OH
PRECIPITATION
NH4VO3 PRECIPITATE
CALCINING
V2O5 PRODUCT

Figure A-16

ARKANSAS VANADIUM PROCESS FLOWSHEET

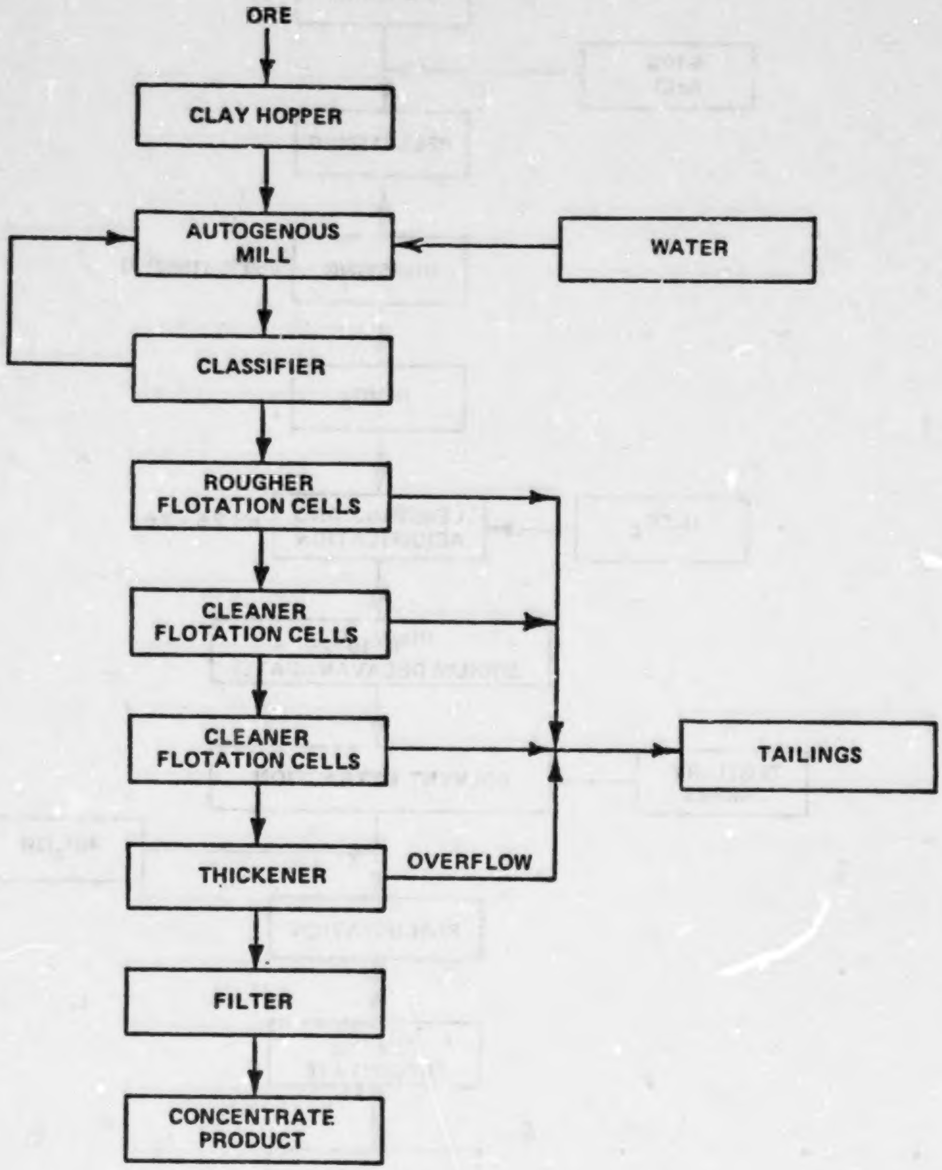

ORE
CLAY HOPPER
AUTOGENOUS MILL
WATER
CLASSIFIER
ROUGHER FLOTATION CELLS
CLEANER FLOTATION CELLS
CLEANER FLOTATION CELLS
TAILINGS
THICKENER
OVERFLOW
FILTER
CONCENTRATE PRODUCT

Figure A-17

FLOW DIAGRAM FOR BENEFICIATION OF MERCURY ORE BY FLOTATION

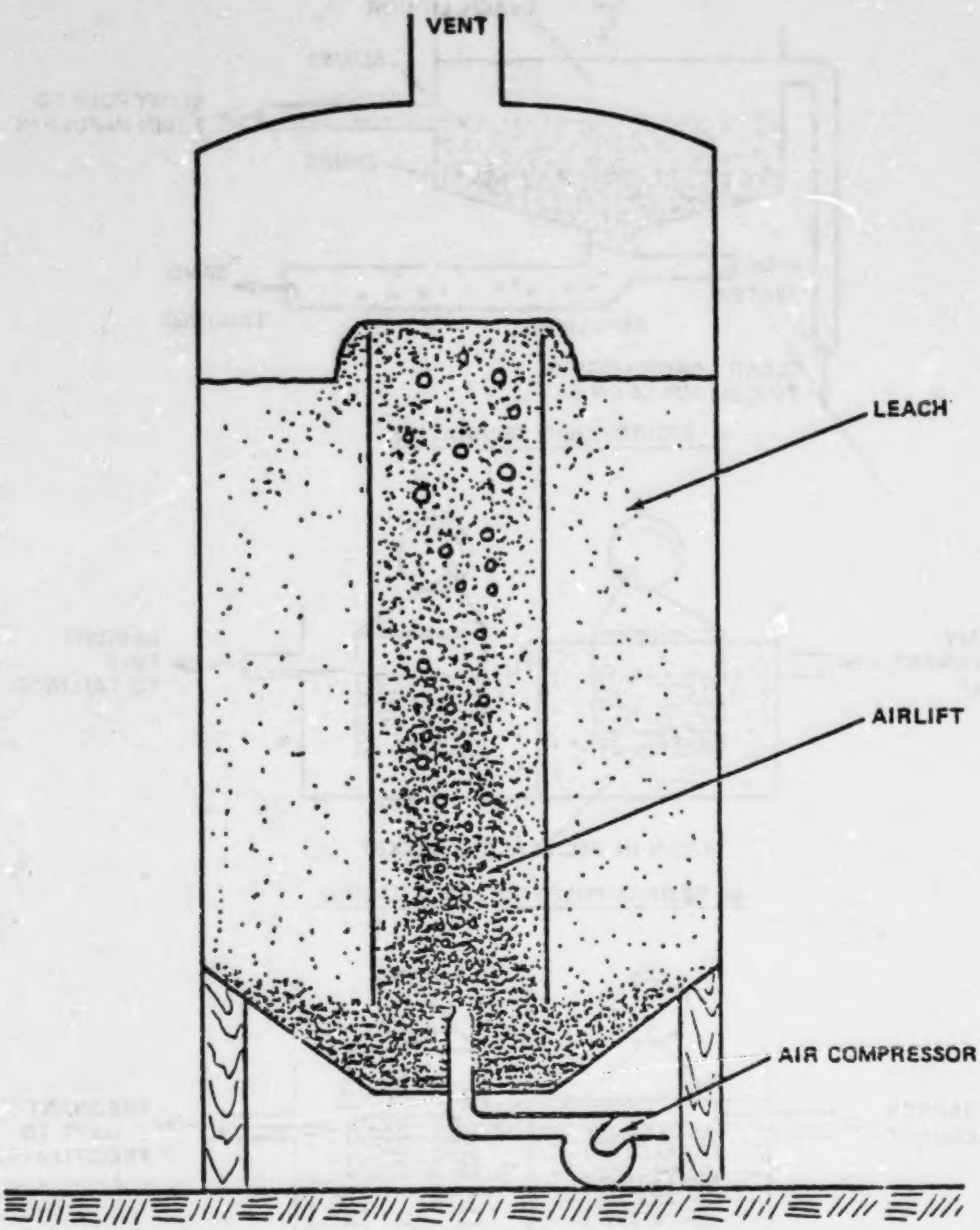
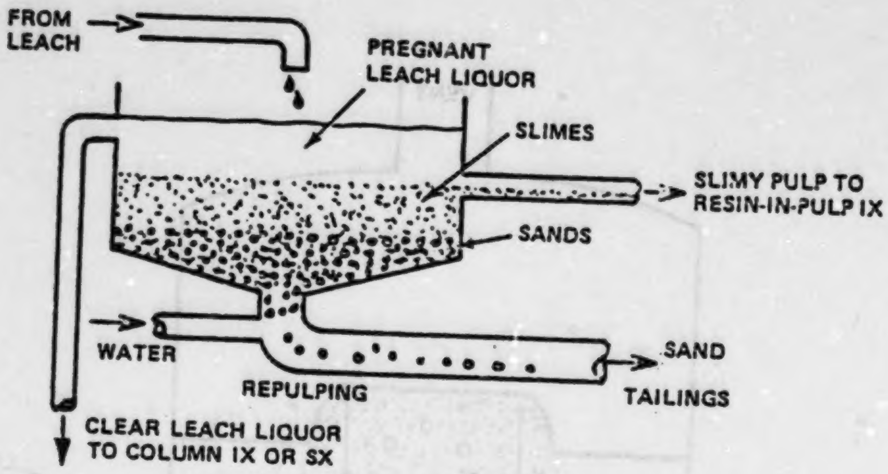
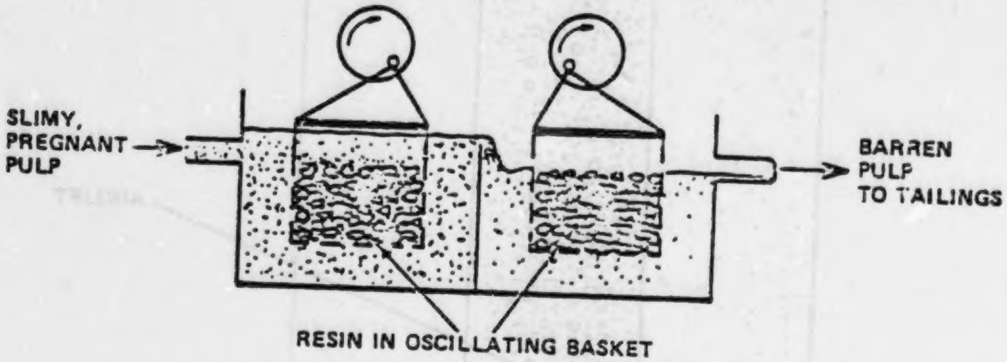

VENT
LEACH
AIRLIFT
AIR COMPRESSOR

Figure A-18

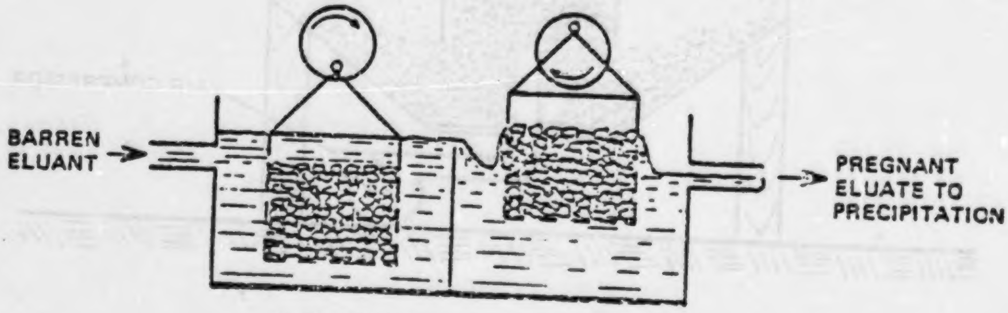
PACHUCA TANK FOR ALKALINE LEACHING


FROM LEACH
PREGNANT LEACH LIQUOR
SLIMES
SLIMY PULP TO RESIN-IN-PULP IX
SANDS
WATER
SAND
REPULPING
TAILINGS
CLEAR LEACH LIQUOR TO COLUMN IX OR SX

a) LIQUID/SOLID SEPARATION


SLIMY, PREGNANT PULP
BARREN PULP TO TAILINGS
RESIN IN OSCILLATING BASKET

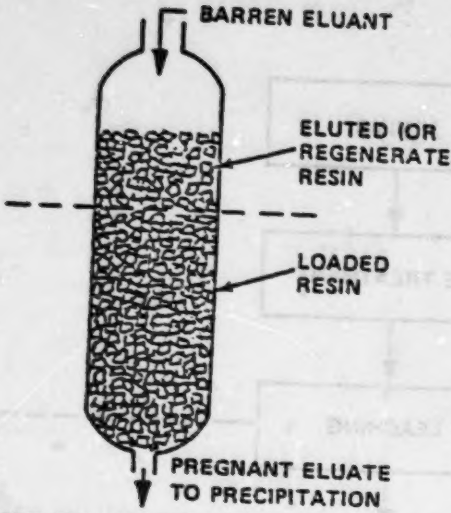
b) RESIN-IN-PULP PROCESS: LOADING


BARREN ELUANT
PREGNANT ELUATE TO PRECIPITATION

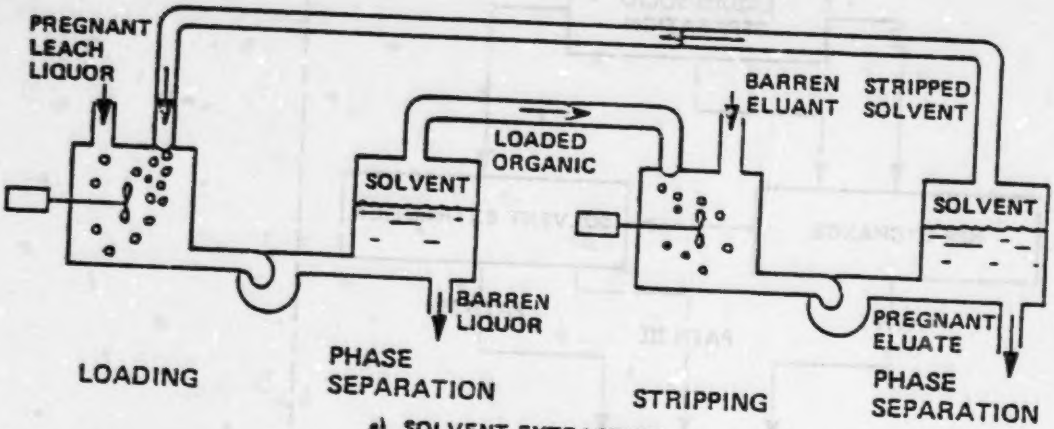
c) RESIN-IN-PULP PROCESS: ELUTING

Figure A-19

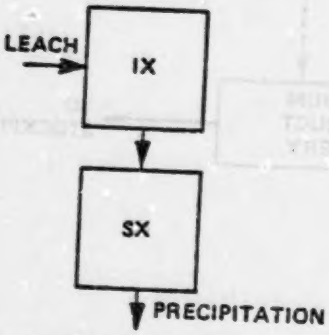
CONCENTRATION PROCESSES AND TERMINOLOGY


BARREN ELUANT
ELUTED (OR REGENERATED) RESIN
LOADED RESIN
PREGNANT ELUATE TO PRECIPITATION

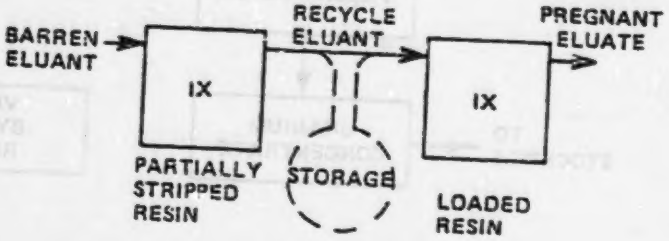
d) FIXED-BED COLUMN ION EXCHANGE/ELUTION


PREGNANT LEACH LIQUOR
SOLVENT
LOADED ORGANIC
BARREN ELUANT
STRIPPED SOLVENT
SOLVENT
BARREN LIQUOR
PREGNANT ELUATE
LOADING
PHASE SEPARATION
STRIPPING
PHASE SEPARATION

e) SOLVENT EXTRACTION


LEACH
IX
SX
PRECIPITATION

f) ELUEX PROCESS


BARREN ELUANT
IX
RECYCLE ELUANT
IX
PREGNANT ELUATE
PARTIALLY STRIPPED RESIN
STORAGE
LOADED RESIN

g) SPLIT ELUTION

Figure A-19
CONCENTRATION PROCESSES AND TERMINOLOGY (Continued)

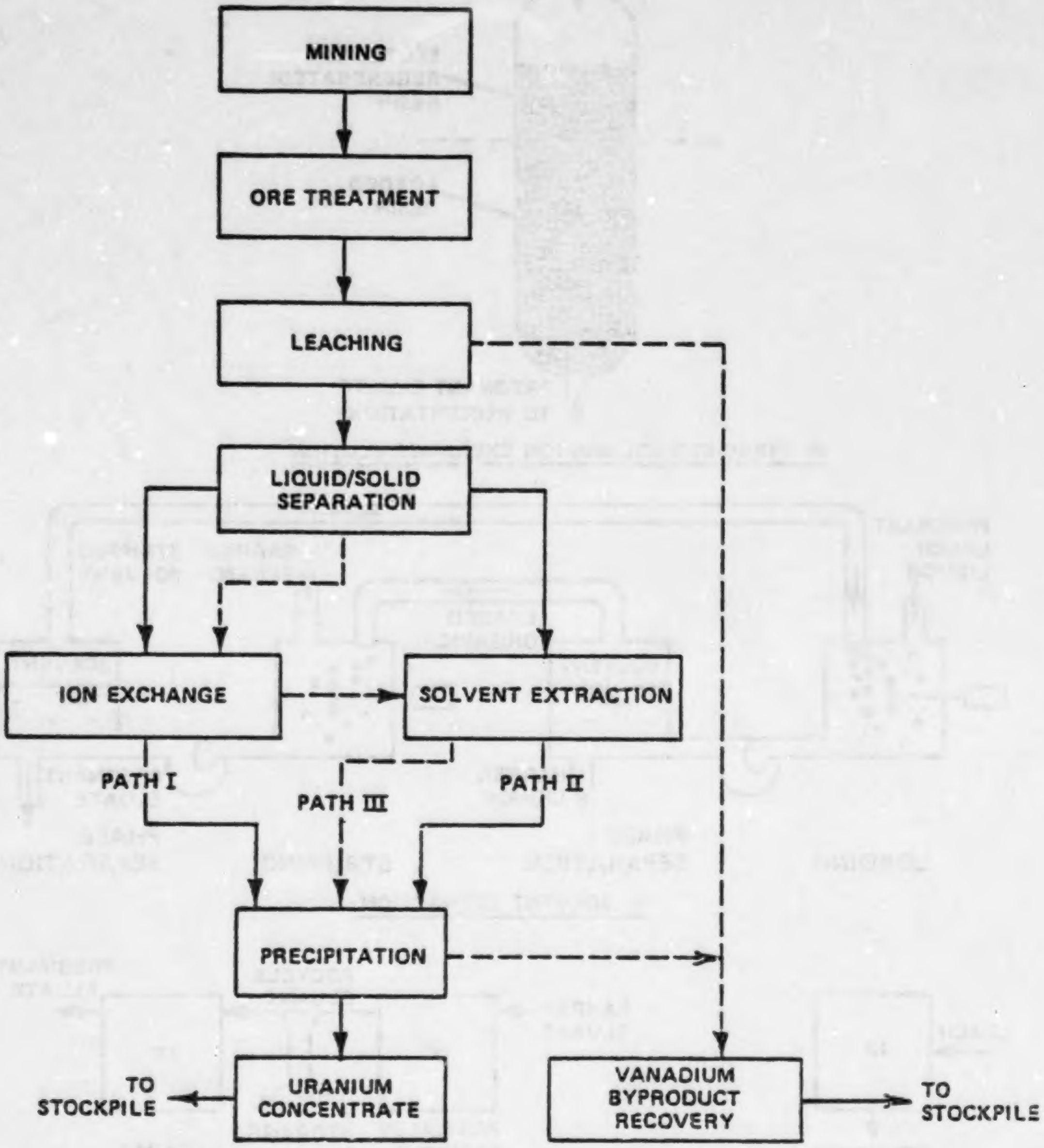

MINING
ORE TREATMENT
LEACHING
LIQUID/SOLID SEPARATION
ION EXCHANGE
SOLVENT EXTRACTION
PATH I
PATH III
PATH II
PRECIPITATION
TO STOCKPILE
URANIUM CONCENTRATE
VANADIUM BYPRODUCT RECOVERY
TO STOCKPILE

Figure A-20

GENERALIZED FLOW DIAGRAM FOR PRODUCTION OR URANIUM VANADIUM, AND RADIUM

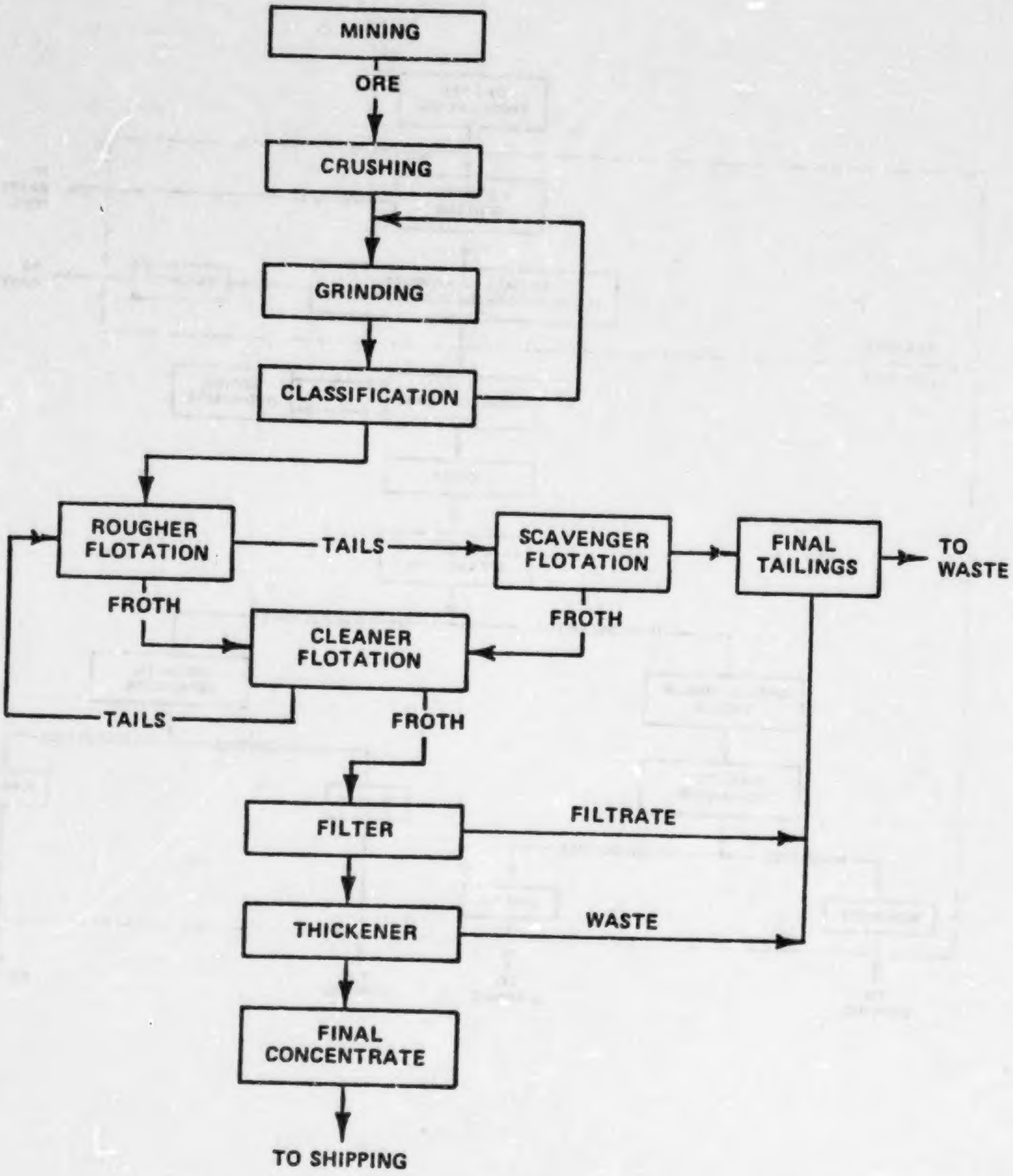

MINING
ORE
CRUSHING
GRINDING
CLASSIFICATION
ROUGHER FLOTATION
TAILS
SCAVENGER FLOTATION
FINAL TAILINGS
TO WASTE
FROTH
CLEANER FLOTATION
FROTH
TAILS
FROTH
FILTER
FILTRATE
THICKENER
WASTE
FINAL CONCENTRATE
TO SHIPPING

Figure A-21

BENEFICIATION OF ANTIMONY SULFIDE ORE BY FLOTATION

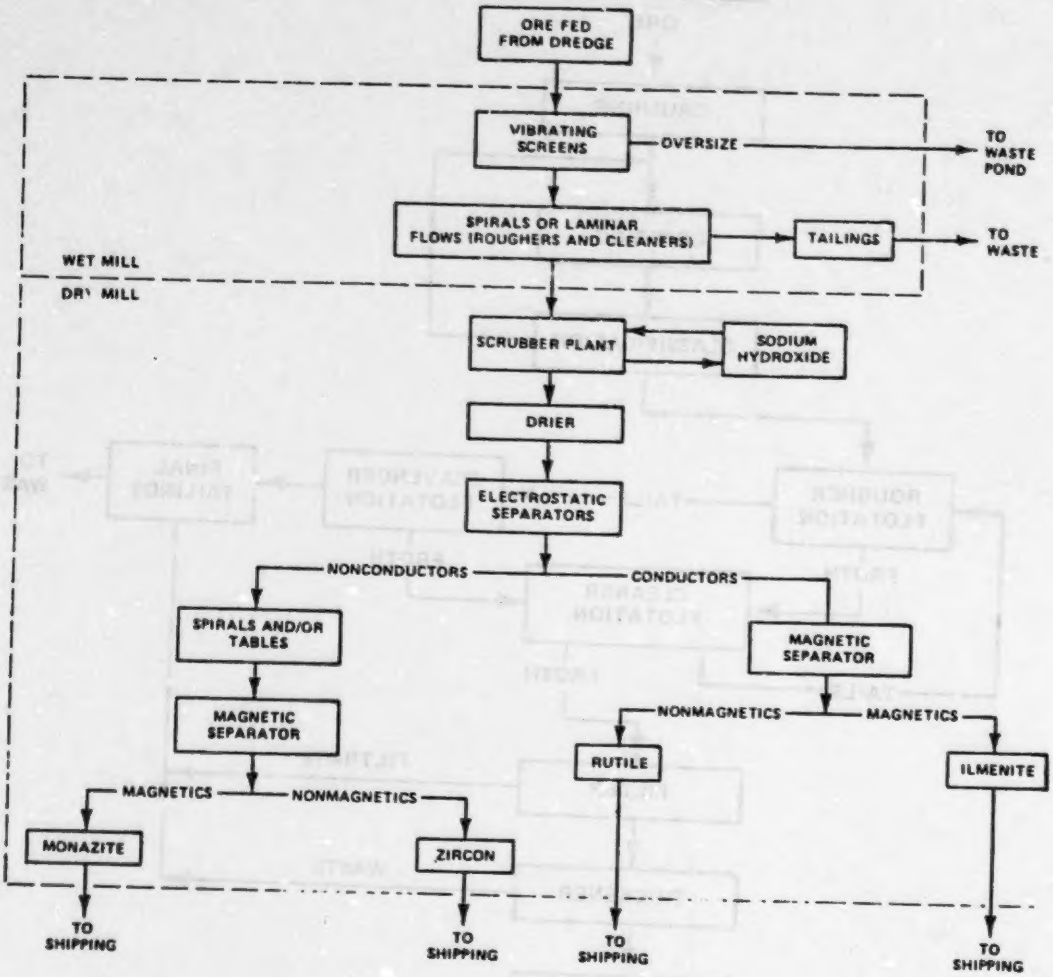

ORE FED FROM DREDGE
VIBRATING SCREENS
OVERSIZE
TO WASTE POND
SPIRALS OR LAMINAR FLOWS (ROUGHERS AND CLEANERS)
TAILINGS
TO WASTE
WET MILL
DRY MILL
SCRUBBER PLANT
SODIUM HYDROXIDE
DRIER
ELECTROSTATIC SEPARATORS
NONCONDUCTORS
CONDUCTORS
SPIRALS AND/OR TABLES
MAGNETIC SEPARATOR
MAGNETIC SEPARATOR
NONMAGNETICS
MAGNETICS
RUTILE
ILMENITE
MAGNETICS
NONMAGNETICS
MONAZITE
ZIRCON
TO SHIPPING
TO SHIPPING
TO SHIPPING
TO SHIPPING

Figure A-22

BENEFICIATION OF HEAVY MINERAL BEACH SANDS

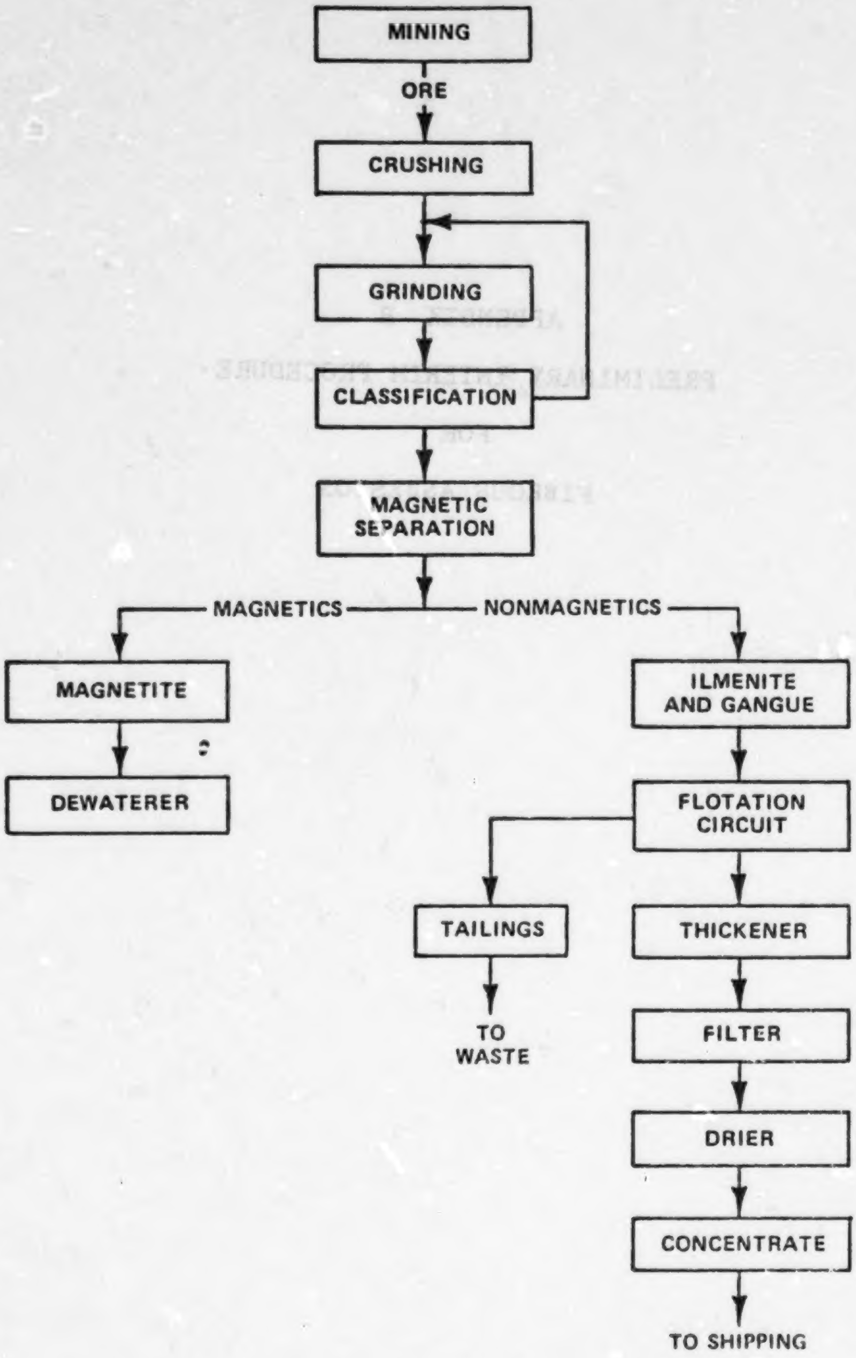

MINING
ORE
CRUSHING
GRINDING
CLASSIFICATION
MAGNETIC SEPARATION
MAGNETICS
NONMAGNETICS
MAGNETITE
DEWATERER
ILMENITE AND GANGUE
FLOTATION CIRCUIT
TAILINGS
TO WASTE
THICKENER
FILTER
DRIER
CONCENTRATE
TO SHIPPING

Figure A-23
BENEFICIATION OF ILMENITE MINED FROM A ROCK DEPOSIT

APPENDIX B

PRELIMINARY INTERIM PROCEDURE FOR FIBROUS ASBESTOS

19 JAN 1979

DRAFT

INTERIM METHOD
FOR ASBESTOS
IN WATER

by

Charles H. Anderson and J. MacArthur Long

Revised December 29, 1978

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Athens, Georgia 30605

Preface to Revised EPA Interim Method for Determining Asbestos in Water

In July 1976 the Preliminary Interim Method for Determining Asbestos in Water was issued by the Athens Environmental Research Laboratory. That method was perceived as representing the current state-of-the-art in asbestos analytical methodology. The objective of writing the method was to present a procedure analytical laboratories could follow that would result in a better agreement of analytical results. In the past two years, a significant amount of additional experimental work has generated data that provide the basis for a more definitive method than was possible previously.

This revised Interim Method reflects the improvements that have been made in asbestos analytical methodology since the initial procedure was drafted. The general approach to the analytical determination, however, remains the same as previously outlined. That is, asbestos fibers are separated from water by filtration on a sub-micron pore size membrane filter. The asbestos fibers are then counted, after dissolving the filter material, by direct observation in a transmission electron microscope.

The major change in the initial procedure is the elimination of the condensation washer as a means of sample preparation. Intra- and inter-laboratory precision data for the method are presented. Also, a suggested statistical evaluation of grid fiber counts is included.

DRAFT

ASBESTOS

(Interim Method)

(Transmission Electron Microscopy Method)

1. Scope and Application

1.1 This method is applicable to drinking water and water supplies.

1.2 The method determines the number of asbestos fibers/liter, their size (length and width), the size distribution, and total mass. The method distinguishes chrysotile from amphibole asbestos. The detection limits are variable and depend upon the amount of total extraneous particulate matter in the sample as well as the contamination level in the laboratory environment. Under favorable circumstances 0.1 MFL (million fibers per liter) can be detected. The detection limit for total mass of asbestos fibers is also variable and depends upon the fiber size and size distribution in addition to the factors affecting the total fiber count. The detection limit under favorable conditions is in the order of 0.1 ng/l.

1.3 The method is not intended to furnish a complete characterization of all the fibers in water.

1.4 It is beyond the scope of this method to furnish detailed instruction in electron microscopy, electron diffraction or crystallography. It is assumed that those using this method will be sufficiently knowledgeable in these fields to understand the methodology involved.

2. Summary of Method

2.1 A variable, known volume of water sample is filtered through a membrane filter of sufficiently small pore size to trap asbestos fibers. A small portion of the filter with deposited fibers is placed on an electron microscope grid and the filter material removed by gentle solution in organic solvent. The material remaining on the electron microscope grid is examined in a transmission microscope at high magnification. The asbestos fibers are identified by their morphology and electron diffraction pattern and their length and width are measured. The total area examined in the electron microscope is determined and the number of asbestos fibers in this area is counted. The concentration in MFL (millions of fibers/liter) is calculated from the number of fibers counted, the amount of water filtered, and the ratio of the total filtered area/sampled filter area. The mass/liter is calculated from the assumed density and the volume of the fibers.

3. Definitions

Asbestos - A generic term applied to a variety of commercially useful fibrous silicate minerals of the serpentine or amphibole mineral groups.

Fiber - Any particulate that has parallel sides and a length/width ratio greater than or equal to 3:1.

Aspect Ratio - The ratio of length to width.

Chrysotile - A nearly pure hydrated magnesium silicate, the fibrous form of the mineral serpentine, possessing a unique layered structure in which the layers are wrapped in a helical cylindrical manner about the fiber axis.

Amphibole Asbestos - A double chain fibrous silicate mineral consisting of Si_4O_{11} units, laterally linked by various cations such as aluminum, calcium, iron, magnesium, and sodium. The members of the amphibole asbestos consist of the following: crocidolite, cummingtonite-gruenerite, and the fibrous forms of tremolite, actinolite and anthophyllite. These minerals consist of or contain fibers formed through natural growth processes. Mineral fragments that conform to the definition of a fiber and that are formed through a crushing and milling process are analytically indistinguishable from the naturally formed fibers by this method.

Detection Limit - The calculated concentration in MFL, equivalent to one fiber above the background or blank count. (Section 8.6).

Statistically Significant - Any concentration based upon a total fiber count of five or more in 20 grid squares.

4. Sample Handling and Preservation

4.A Sampling

It is beyond the scope of this procedure to furnish detailed instructions for field sampling; the general principles of sampling waters are applicable. There are some considerations that apply to asbestos fibers, a special type of particulate matter. These fibers are

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small, and in water range in length from .1 μm to 20 μm or more. Because of the range of size there may be a vertical distribution of particle sizes. This distribution will vary with depth depending upon the vertical distribution of temperature as well as the local meteorological conditions. Sampling should take place according to the objective of the analysis. If a representative sample of a water supply is required a carefully designated set of samples should be taken representing the vertical as well as the horizontal distribution and these samples composited for analysis.

4.1 Containment Vessel

The sampling container shall be a clean conventional polyethylene, screw-capped bottle capable of holding at least one liter. The bottle should be rinsed at least two times with the water that is being sampled prior to sampling.

NOTE: Glass vessels are not suitable as sampling containers.

4.2 Quantity of Sample

A minimum of approximately one liter of water is required and the sampling container should not be filled. It is desirable to obtain two samples from one location.

4.3 Sample Preservation

No preservatives should be added during sampling and the addition of acids should be particularly avoided. If the sample cannot be filtered in the laboratory within 48 hours of its arrival, sufficient amounts (1 ml/l of sample) of a 2.71% solution of mercuric chloride to give a final concentration of 20 ppm of Hg may be added to prevent bacterial growth.

NOTE 1: It has been reported that prevention of bacterial growth in water samples can be achieved by storing the samples in the dark.

5. Interferences

5.1 Misidentification

The guidelines set forth in this method for counting fibrous asbestos require a positive identification by both morphology and crystal structure as shown by an electron diffraction pattern. Chrysotile asbestos has a unique tubular structure, usually showing the presence of a central canal, and exhibits a unique characteristic electron diffraction pattern. Although halloysite fibers may show a streaking similar to chrysotile they do not exhibit its characteristic triple set of double spots or 5.3A layer line. It is highly improbable that a non-asbestiform fiber would exhibit the distinguishing chrysotile features. Although amphibole fibers exhibit characteristic morphology and election diffraction patterns, they do not have the unique properties exhibited by chrysotile. It is therefore possible though not probable for misidentification to take place. Hornblende is an amphibole and, in a fibrous form, will be mistakenly identified as amphibole asbestos.

It is important to recognize that a significant variable fraction of both chrysotile and amphibole asbestos fibers do not exhibit the required confirmatory electron diffraction pattern. This absence of diffraction is attributable to unfavorable fiber orientation and fiber sizes. The results reported will therefore be low as compared to the absolute number of asbestos fibers that are present.

5.2 Obscuration

If there are large amounts of organic or amorphous inorganic materials present, some small asbestos fibers may not be observed because of physical overlapping or complete obscuration. This will result in low values for the reported asbestos content.

5.3 Contamination

Although contamination is not strictly considered an interference, it is an important source of erroneous results, particularly for chrysotile. The possibility of contamination should therefore always be a consideration.

5.4 Freezing

The effect of freezing on asbestos fibers is not known but there is reason to suspect that fiber break down could occur and result in a higher fiber count than was present in the original sample. Therefore the sample should be transported to the laboratory under conditions that would avoid freezing.

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6. Equipment and Apparatus

6.1 Specimen Preparation Laboratory

The ubiquitous nature of asbestos, especially chrysotile, demands that all sample preparation steps be carried out to prevent the contamination of the sample by air-borne or other source of asbestos. The prime requirement of the sample preparation laboratory is that it be sufficiently free from asbestos contamination that a specimen blank determination using 200 ml of asbestos-free water yields no more than 2 fibers in twenty grid squares of a conventional 200 mesh electron microscope grid.

In order to achieve this low level of contamination, the sample preparation area should be a separate conventional clean room facility. The room should be operated under positive pressure and have incorporated electrostatic precipitators in the air supply to the room, or alternatively absolute (HEPA) filters. There should be no asbestos floor or ceiling tiles, transite heat-resistant boards, nor asbestos insulation. Work surfaces should be stainless steel or Formica or equivalent. A laminar flow hood should be provided for sample manipulation. Disposable plastic lab coats and disposable overshoes are recommended. Alternatively new shoes for all operators should be provided and retained for clean room use only. A mat (Tacky Mat, Liberty Industries, 589 Deming Rd., Berlin, Connecticut 06037, or equivalent) should be placed inside the entrance to the room to trap any gross contamination inadvertently brought into the room from contaminated shoes. Normal electrical and water services, including a distilled water supply should be provided. In addition a source of ultra-pure water from a still or filtration-ion exchange system is desirable.

6.2 Instrumentation

6.2.1 Transmission Electron Microscope. A transmission electron microscope that operates at a minimum of 80 KV, has a resolution of 1.0 nm and a magnification range of 300 to 100,000. If the upper limit is not attainable directly it may be attained through the use of auxiliary optical viewing. It is mandatory that the instrument be capable of carrying out selected area electron diffraction (SAED) on an area of 300 nm^2. The viewing screen shall have either a millimeter scale, concentric circles of known radii, or other devices to measure the length and width of the fiber. Most modern transmission microscopes meet the requirements for magnification and resolution.

An energy-dispersive X-ray spectrometer is useful for the identification of suspected asbestiform minerals; this accessory to the microscope, however, is not mandatory.

6.2.2 Data Processor. The large number of repetitive calculations make it convenient to use computer facilities together with relatively simple computer programs.

6.2.3 Vacuum Evaporator. For depositing a layer of carbon on the Nuclepore filter, and for preparing carbon coated grids.

6.2.4 Low Temperature Plasma Asher. To be used for the removal of organic material (including the filter) from samples containing so much organic matter that asbestos fibers are obscured. The sample chamber should be at least 10-cm diameter.

6.3 Apparatus, Supplies and Reagents

6.3.1 Jaffe Wick Washer. For dissolving Nuclepore filter. Assemble as in 8.3.1. It is illustrated in Figure 1.

6.3.2 Filtering Apparatus. 47-mm funnel (Cat No. XX1504700, Millipore Corporation, Order Service Dept., Bedford, MA 01730). Used to filter water samples. 25-mm funnel (Millipore Cat No. XX1002500). Used to filter dispersed ash samples.

6.3.3 Vacuum Pump. For use in sample filtration. Should provide vacuum up to 20 inches of mercury.

6.3.4 EM Grids. 200-mesh copper or nickel grids, covered with formvar film for use with Nuclepore-Jaffe sample preparation method. These grids may be purchased from manufacturers of electron microscopic supplies or prepared by standard electron microscopic grid preparation procedures. Finder grids may be substituted and are useful if the re-examination of a specific grid opening is desired.

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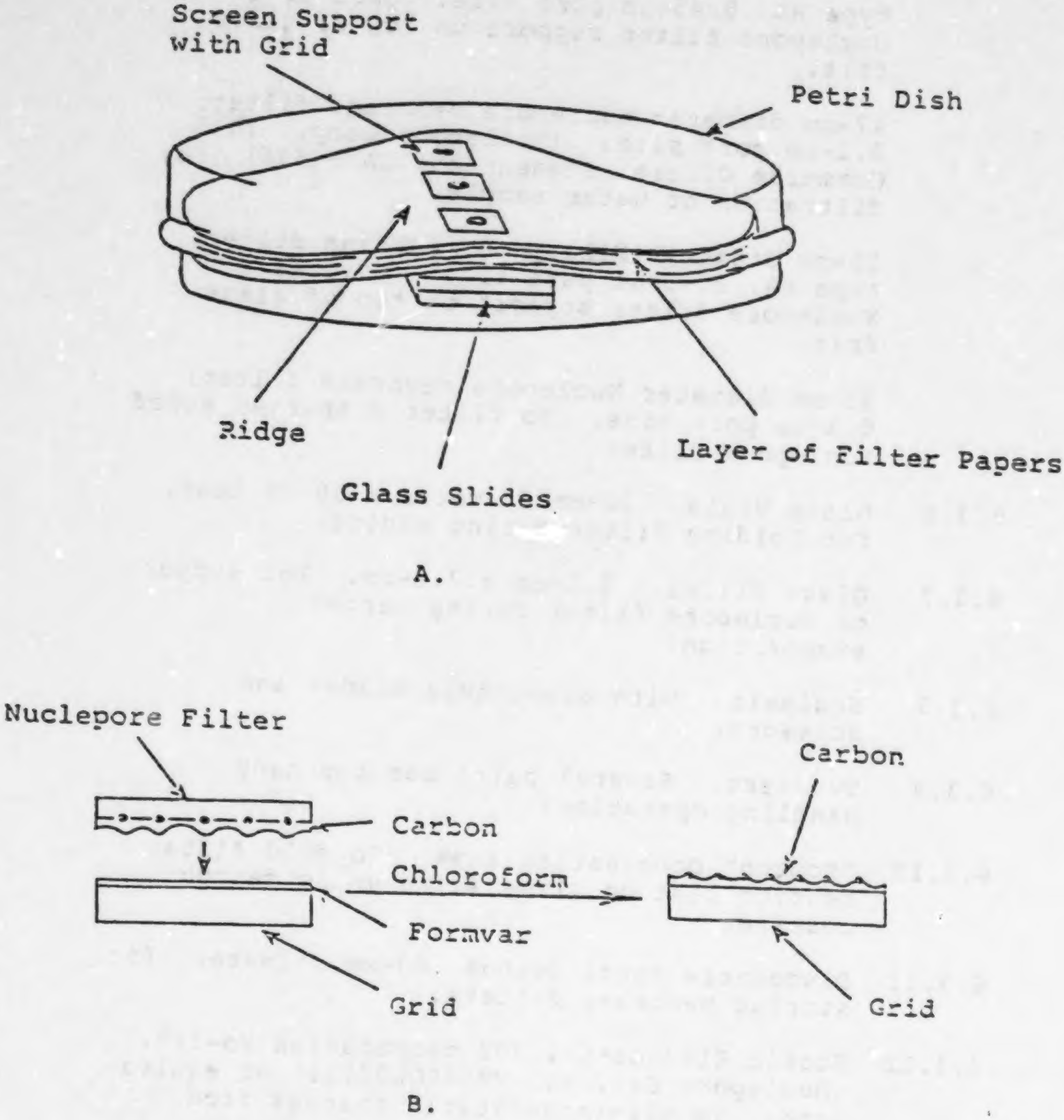
Screen Support with Grid
Petri Dish
Ridge
Layer of Filter Papers
Glass Slides
A.
Nuclepore Filter
Carbon
Carbon
Chloroform
Formvar
Grid
Grid
B.

Figure 1. Modified Jaffe Wick Method
A. Washing Apparatus
B. Washing Process

6.3.5 Membrane Filters.

47-mm diameter Millipore membrane filter, type HA, 0.45-μm pore size. Used as a Nuclepore filter support on top of glass frit.

47-mm diameter Nuclepore membrane filter; 0.1-μm pore size. (Nuclepore Corp., 7035 Commerce Circle, Pleasanton, CA 94566) For filtration of water sample.

25-mm diameter Millipore membrane filter, type HA; 0.45-μm pore size. Used as Nuclepore filter support on top of glass frit.

25-mm diameter Nuclepore membrane filter; 0.1-μm pore size. To filter dispersed ashed Nuclepore filter.

6.3.6 Glass Vials. 30-mm diameter x 80-mm long. For holding filter during ashing.

6.3.7 Glass Slides. 5.1-cm x 7.5-cm. For support of Nuclepore filter during carbon evaporation.

6.3.8 Scalpels. With disposable blades and scissors.

6.3.9 Tweezers. Several pairs for the many handling operations.

6.3.10 "Scotch" Doublestick tape. To hold filter section flat on glass slide while carbon coating.

6.3.11 Disposable Petri dishes, 50-mm diameter, for storing membrane filters.

6.3.12 Static Eliminator, 500 microcuries Po-210. (Nuclepore Cat. No. V090POL00101) or equivalent. To eliminate static charges from membrane filters.

6.3.13 Carbon rods, spectrochemically pure, 1.8" dia., 3.6 mm x 1.0 mm neck. For carbon coating.

6.3.14 Carbon rod sharpener. (Cat. No. 1204. Ernest F. Fullam, Inc., P. O. Box 444, Schenectady, NY 12301) For sharpening carbon rods to a neck of specified length and diameter.

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6.3.15 Ultrasonic Bath. (50 watts, 55 HKz). For dispersing ashed sample and for general cleaning.

6.3.16 Graduated Cylinder, 500 ml.

6.3.17 Spot plate.

6.3.18 10-µl Microsyringe. For administering drop of solvent to filter section during sample preparation.

6.3.19 Carbon grating replica, 2160 lines/mm. For calibration of EM magnification.

6.3.20 Filter paper. S & S #589 Black Ribbon (9-cm circles) or equivalent absorbent filter paper. For preparing Jaffe Wick Washer.

6.3.21 Screen supports (copper or stainless steel) 12 mm x 12 mm, 200 mesh. To support specimen grid in Jaffe Wick Washer.

6.3.22 Chloroform, spectro grade, doubly distilled. For dissolving Nuclepore filters.

6.3.23 Asbestos. Chrysotile (Canadian), Crocidolite, Amosite. UICC (Union Internationale Contre le Cancer) Standards. Available from Duke Standards Company, 445 Sherman Avenue, Palo Alto, CA 94306.

6.3.24 Petri dish, glass (100 mm diameter x 15 mm high). For modified Jaffe Wick Washer.

6.3.25 Alconox. (Alconox, Inc., New York, NY 10003) For cleaning glassware. Add 7.5 g Alconox to a liter of distilled water.

6.3.26 Parafilm. (American Can Company, Neenah, WI) Use as protective covering for clean glassware.

6.3.27 Pipets, disposable, 5 ml and 50 ml.

6.3.28 Distilled or deionized water. Filter through 0.1-µm Nuclepore filter for making up all reagents and for final rinsing of glassware, and for preparing blanks.

6.3.29 Mercuric chloride, 2.71% solution w/v. Used as sample preservative. See 4.3. Add 5.42 g of reagent grade mercuric chloride ($HgCl_2$)

to 100 ml distilled water and dissolve by shaking. Dilute to 200 ml with additional water. Filter through 0.1-μm Nuclepore filter paper before using.

7. Preparation of Standards

Reference standard samples of asbestos that can be used for quality control for a quantitative analytical method are not available. It is, however, necessary for each laboratory to prepare at least two suspensions, one of chrysotile and another of a representative amphibole. These suspensions can then be used for intra-laboratory control and furnish standard morphology photographs and diffraction patterns.

7.1 Chrysotile Stock Solution.

Grind about 0.1 g of UICC chrysotile in an agate mortar for several minutes, or until it appears to be a powder. Weigh out 10 mg and transfer to a clean 1 liter volumetric flask, add several hundred ml of filtered distilled water containing one ml of a stock mercuric chloride solution and then make up to 1 liter with filtered distilled water. To prepare a working solution, transfer 10 ml of the above suspension to another 1-liter flask, add 1 ml of a stock mercuric chloride solution and make up to 1 liter with filtered distilled water. This suspension contains 100 μg per liter. Finally transfer 1 ml of this suspension to a 1-liter flask, add 1 ml of a stock mercuric chloride solution and make up to volume with filtered distilled water. The final suspension will contain 5-10 MFL and is suitable for laboratory testing.

7.2 Amphibole Stock Dispersion.

Prepare amphibole suspensions from UICC amphibole samples as in Section 7.1.

7.3 Identification Standards.

Prepare electron microscopic grids containing the UICC asbestos fibers according to 8, Procedure, and obtain representative photographs of each fiber type and its diffraction pattern for future reference.

8. Procedure

8.1 Filtration.

The separation of the insoluble material, including asbestiform minerals, through filtration and subse-

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quent deposition on a membrane filter is a very critical step in the procedure. The objective of the filtration is not only to separate, but also to distribute uniformly the particulate matter such that discreet particles are deposited with a minimum of overlap.

The volume filtered will range from 50-500 ml. In an unknown sample the volume can not be specified in advance because of the presence of variable amounts of particulate matter. In general, sufficient sample is filtered such that a very faint stain can be observed on the filter medium. The maximum loading that can be tolerated is 20 $\mu g/cm^2$, or about 200 μg on a 47-mm diameter filter; 5 $\mu g/cm^2$ is near optimum. If the total solids content is known, an estimate of the maximum volume tolerable can be obtained. In a sample of high solids content, where less than 50 ml is required, the sample should be diluted with filtered distilled water so that a minimum total of 50 ml of water is filtered. This step is necessary to allow the insoluble material to deposit uniformly on the filter. The filtration funnel assembly must be scrupulously clean and cleaned before each filtration. The filtration should be carried out in a laminar flow hood.

NOTE 1: The following cleaning procedure has been found to be satisfactory:

Wash each piece of glassware three times with distilled water. Following manufacturer's recommendations use the ultrasonic bath with an Alconox-water solution to clean all glassware. After the ultrasonic cleaning rinse each piece of glassware three times with distilled water. Then rinse each piece three times with deionized water which has been filtered through 0.1-μm Nuclepore filter. Dry in an asbestos-free oven. After the glassware is dry, seal openings with parafilm.

8.1.1 Filtration

a. Assemble the vacuum filtration apparatus incorporating the .1-μm Nuclepore backed with 0.45-μm Millipore filter. See 8.3.2.

b. Vigorously agitate the water sample in its container.

c. If the required filtration volume can be estimated, either from turbidity estimates of suspended solids or previous experience, immediately withdraw the proper volume from the container and add the entire volume to the 47-mm diameter funnel. Apply vacuum sufficient for filtration but gentle enough to avoid the formation of a vortex. If a completely unknown sample is being analyzed, a slightly modified procedure must be followed. Pour 500 ml of a well-mixed sample into a 500-ml graduated cylinder and immediately transfer the entire contents to the prepared vacuum filtration apparatus. Apply vacuum gently and continue suction until all of the water has passed through the filter. If the resulting filter appears obviously coated or discolored, it is recommended that another filter be prepared in the same manner, but this time using only 200 or 100 ml of sample.

NOTE 1: Do not add more water after filtration has started and do not rinse the sides of the funnel.

NOTE 2: Nuclepore filter is basically a hydrophobic material. The manufacturer applies a detergent to the surface of the filter in order to render it hydrophilic; this process, however, does not appear to be entirely satisfactory in some batches. Pretreatment of the filter in a low temperature asher at 10 watts for 10 seconds can be used to render the surface of the filter hydrophilic. This process will significantly decrease the islands of sparse deposit frequently observed.

d. Disassemble the funnel, remove the filter and dry in a covered petri dish.

8.2 Preparation of Electron Microscope Grids.

The preparation of the grid for examination in the microscope is a critical step in the analytical procedure. The objective is to remove the organic filter material from the asbestos fibers with a minimum loss and movement and with a minimum breakage of the grid support film.

If the sample contains organic matter in such amounts that interfere with fiber counting and identification a preliminary ashing step is required. See 8.5.

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8.3 Nuclepore Filter, Modified Jaffe Wick Technique.

8.3.1 Preparation of Modified Jaffe Washer

Place three glass microscope slides (75 mm x 22 mm) one on top of the other in a petri dish (100 mm x 15 mm) along a diameter. Place 14 S & S #589 Black Ribbon filter papers (9-cm circles) in the petri dish over the stack of microscope slides. Place three copper mesh screen supports (12 mm x 12 mm) along the ridge formed by the stack of slides underneath the layer of filter papers. Place an EM specimen grid on each of the screen supports. See Fig. 1.

NOTE 1: A stack of 30-40 S & S filters (7-cm circle) can be substituted for the 14 filters and microscope slides in preparing the Jaffe washer.

8.3.2 Vacuum Filtration Unit

Assemble the vacuum filtration unit. Place a 0.45-µm Millipore filter type HA on the glass frit and then position a 0.1-µm Nuclepore filter, shiny side up, on top of the Millipore filter. Apply suction to center the filters flat on the frit. Attach the filter funnel and shut off the suction.

8.3.3 Sample Filtration

See 8.1.1.

8.3.4 Sample Drying

Remove the filter funnel and place the Nuclepore filter in a loosely covered petri dish to dry. The petri dish containing the filter may be placed in an asbestos-free oven at 45° C for 30 minutes to shorten the drying time.

8.3.5 Selection of Section for Carbon Coating

Using a small pair of scissors or sharp scalpel cut out a retangular section of the Nuclepore filter. The minimum approximate dimensions should be 15 mm long and 3 mm wide. Avoid selection near the perimeter of the filtration area.

8.3.6 Carbon Coating the Filter

Tape the two ends of the selected filter section to a glass slide using "Scotch" tape. Take care not to stretch the filter section. Identify the filter section using a china marker on the slide. Place the glass slide with the filter section into the vacuum evaporator. Insert the necked carbon rod and, following manufacturer's instructions, obtain high vacuum. Evaporate the neck, with the filter section rotating, at a distance of approximately 7.5 cm from the filter section to obtain a 30-50 nm layer of carbon on the filter paper. Evaporate the carbon in several short bursts rather than continuously to prevent overheating the surface of the Nuclepore filter.

NOTE 1: Overheating the surface tends to crosslink the plastic, rendering the filter dissolution in chloroform difficult.

NOTE 2: The thickness of the carbon film can be monitored by placing a drop of oil on a porcelain chip that is placed at the same distance from the carbon electrodes as the specimen. Carbon is not visible in the region of the oil drop thereby enabling the visual estimate of the deposit thickness by the contrast differential.

8.3.7 Grid Transfer

Remove the filter from the vacuum evaporator and cut out three sections somewhat less than 3 mm x 3 mm and such that the square of Nuclepore fits within the circumference of the grid. Pass each of the filter sections over a static eliminator and then place each of the three sections carbon-side down on separate specimen grids previously placed in the modified Jaffe Washer. Using a microsyringe, place a 10-μl drop of chloroform on each filter section resting on a grid and then saturate the filter pad until pooling of the solvent occurs below the ridge formed by the glass slides inserted under the layer of filter papers. Place the cover on the petri dish and allow the grids to remain in the washer for approximately 24 hours. Do not allow the chloroform to completely evaporate before the grids are removed. To remove the grids from the washer lift the

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screen support with the grid resting upon it and set this in a spot plate depression to allow evaporation of any solvent adhering to the grid. The grid is now ready for analysis or storage.

8.4 Electron Microscopic Examination

8.4.1 Microscope Alignment and Magnification Calibration

Following the manufacturer's recommendations carry out the necessary alignment procedures for optimum specimen examination in the electron microscope. Calibrate the routinely used magnifications using a carbon grating replica.

NOTE 1: Screen magnification is not necessarily equivalent to plate magnification.

8.4.2 Grid Preparation Acceptability

After inserting the specimen into the microscope adjust the magnification low enough (300X-1000X) to permit viewing complete grid squares. Inspect at least 10 grid squares for fiber loading and distribution, debris contamination, and carbon film continuity.

Reject the grid for counting if:

1) The grid is too heavily loaded with fibers to perform accurate counting and diffraction operations. A new sample preparation either from a smaller volume of water or from a dilution with filtered distilled water must then be prepared.

2) The fiber distribution is noticeably uneven. A new sample preparation is required.

3) The debris contamination is too severe to perform accurate counting and diffraction operations. If the debris is largely organic the filter must be ashed and redispersed (see 8.3). If inorganic the sample must be diluted and again prepared.

4) The majority of grid squares examined have broken carbon films. A different grid

preparation from the same initial filtration must be substituted.

8.4.3 Procedure for Fiber Counting

There are two methods commonly used for fiber counting. In one method (A) 100 fibers, contained in randomly selected fields of view, are counted. The number of fields plus the area of a field of view must be known when using this method. In the other method (B), all fibers (at least 100) in several grid squares or 20 grid squares are counted. The number of grid squares counted and the average area of one grid square must be known when using this method.

NOTE 1: The method to use is dependent upon the fiber loading on the grid and it is left to the judgement of the analyst to select the optimum method. The following guidelines can be used: If it is estimated that a grid square (80 µm x 80 µm) contains 50-100 fibers at a screen magnification of 20000X it is convenient to use the field-of-view counting method. If the estimate is less than 50, the grid square method of counting should be chosen. On the other hand, if the fiber count is estimated to be over 300 fibers per grid square, a new grid containing less fibers must be prepared (through dilution or filtration of a smaller volume of water).

8.4.3A Field-of-View Method

After determining that a fiber count can be obtained using this method adjust the screen magnification to 15,-20,000X. Select a number of grid squares that would be as representative as possible of the entire analyzable grid surface. From each of these squares select a sufficient number of fields of view for fiber counting. The number of fields of view per grid square is dependent upon the fiber loading. If more than one field of view per grid square is selected, scan the grid opening orthogonally in an arbitrary pattern which prevents overlapping of fields of view. Carry out the analysis by counting, measuring and identifying (see 8.4.4) approximately 50 fibers on each of two grids.

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The following rules should be followed when using the field of view method of fiber counting. Although these rules were derived for a circular field of view they can be modified to apply to square or rectangular designs.

1) Count all fibers contained within the counting area and not touching the circumference of the circle.

2) Designate the upper right-hand quadrant as I and number in clockwise order. Count all fibers touching or intersecting the arc of quadrants I or IV. Do not count fibers touching or intersecting the arc of quadrants II or III.

3) If a fiber intersects the arc of both quadrants III and IV or I and II count it only if the greater length was outside the arc of quadrants IV and I, respectively.

4) Count fibers intersecting the arc of both quadrants I and III but not those intersecting the arc of both II and IV.

These rules are illustrated in Fig. 2.

8.4.3B Grid Square Method

After determining that a fiber count can be obtained using this method adjust the screen magnification to 15,-20,000X. Position the grid square so that scanning can be started at the left upper corner of the grid square. While carefully examining the grid, scan left to right, parallel to the upper grid bar. When the perimeter of the grid square is reached adjust the field of view down one field width and scan in the opposite direction. The tilting section of the fluorescent screen may be used conveniently as the field of view. Examine the square until all the area has been covered. The analysis should be carried out by counting, measuring and identifying (see 8.4.4) approximately 50 fibers on each of two grids or until 10 grid squares on each of two grids have been counted. Do not count fibers intersecting a grid bar.

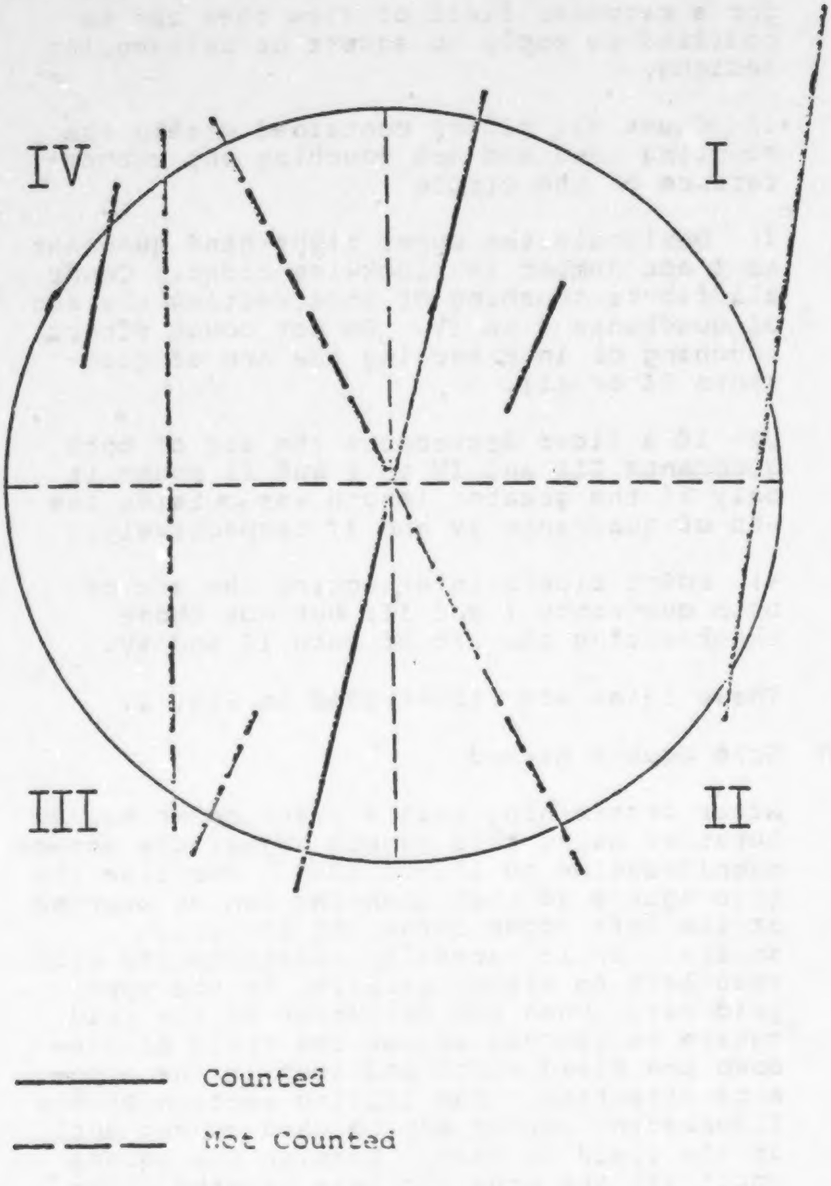
IV
I
III
II
Counted
Not Counted

Figure 2. Illustration of Counting Rules for Field-of-View Method

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8.4.4 Measurement and Identification

Measure and record the length and width of each fiber having an aspect ratio greater than or equal to three. Disregard obvious biological, bacteriological fibers and diatom fragments. Examine the morphology of each fiber using optical viewing if necessary. Tentatively identify, by reference to the UICC standards, chrysotile or possible amphibole asbestos. Attempt to obtain a diffraction pattern of each fiber utilizing the shortest camera length possible. Move the suspected fiber image to the center of the screen and insert a suitable selected area aperture into the electron beam so that the fiber image, or a portion of it, is in the illuminated area. The size of the aperture and the portion of the fiber should be such that particles other than the one to be examined are excluded from the selected area. Observe the diffraction pattern with the 10X binoculars. If an incomplete diffraction pattern is obtained move the particle image around in the selected area to get a clearer diffraction pattern or to eliminate possible interferences from neighboring particles.

Determine whether or not the fiber is chrysotile or an amphibole by comparing the diffraction pattern obtained to the diffraction patterns of known standard asbestos fibers. Confirm the tentative identification of chrysotile and amphibole asbestos from their electron diffraction patterns. Classify each fiber as chrysotile, amphibole, non-asbestos, no diffraction or ambiguous.

NOTE 1: It is convenient to use a tape recorder during the examination of the fibers to record all pertinent data. This information can then be summarized on data sheets or punched cards for subsequent automatic data processing.

NOTE 2: Chrysotile fibers occur as single fibrils, or in bundles. The fibrils generally show a tubular structure with a hollow canal, although the absence of the canal does not rule out its identification. Amphibole asbestos fibers usually exhibit a lath-like structure with irregular ends, but occasionally will resemble chrysotile in appearance.

NOTE 3: The positive identification of asbestos by electron diffraction requires some judgment on the part of the analyst because some fibers give only partial patterns. Chrysotile shows unique prominent streaks on the layer lines nearest the central one and a triple set of double spots on the second layer line. The streaks and the set of double spots are the distinguishing characteristics of chrysotile required for identification. Amphibole asbestos requires a more complete diffraction pattern to be positively identified. As a qualititative guideline, layer lines for amphibole, without the unique streaks (some streaking may be present) of chrysotile, should be present and the arrangement of diffraction spots along the layer lines should be consistent with the amphibole pattern. The pattern should be distinct enough to establish these criteria.

NOTE 4: Chrysotile and thin amphibole fibers may undergo degradation in an electron beam; this is particularly noticeable in small fibers. It may exhibit a pattern for a 1-2 seconds and disappear and the analyst must be alert to note the characteristic features.

NOTE 5: An ambiguous fiber is a fiber that gives a partial electron diffraction pattern resembling asbestos, but insufficient to provide positive identification.

8.4.5 Determination of Grid Square Area

Measure the dimensions of several representative grid squares from each batch of grids with an optical microscope. Calculate the average area of a grid square. This should be done to compensate for variability in grid square dimensions.

8.5 Ashing

Some samples contain sufficiently high levels of organic material that an ashing step is required before fiber identification and counting can be carried out.

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Place the dried Nuclepore filter paper containing the collected sediment into a glass vial (28 mm diameter x 80 mm high). Position the filter such that the filtration side touches the glass wall. Place the vial in an upright position in the low temperature asher. Operate the asher at 50 watts (13.56 MHz) power and 2 psi oxygen pressure. Ash the filter until a thin film of white ash remains. The time required is generally 6 to 8 hours. Allow the ashing chamber to slowly reach atmospheric pressure and remove the vial. Add 10 ml of filtered distilled water to the vial. Place the vial in an ultrasonic bath for 1/2 hour to disperse the ash. Dilute the sample if required.

Assemble the 25-mm diameter filtering apparatus. Center a 25-mm diameter .1-μm Nuclepore filter (with the 0.45-μm Millipore backing) on the glass frit. Apply suction and recenter the filter if necessary. Attach the filter funnel and turn off the suction. Add the water containing the dispersed ash from the vial to the filter funnel. Apply suction and filter the sample. After drying this filter it is ready to be used in preparing sample grids as in 8.3.

NOTE 1: In specifying a 25-mm diameter filter it is assumed that the ashing step is necessary mainly because of the presence of organic material and that the smaller filtering area is desirable from the point of view of concentrating the fibers. If the sample contains mostly inorganic debris such that the smaller filtering area will result in overloading the filter, the 47-mm diameter filter should be used.

NOTE 2: It will be noted that a 10-ml volume is filtered in this case instead of the minimum 50-ml volume specified in 8.1. These volumes are consistent when it is considered that there is approximately a 5-fold difference in effective filtration area between the 25-mm diameter and 47-mm diameter filters.

8.6 Determination of Blank Level

Carry out a blank determination with each batch of samples prepared, but a minimum of one per week. Filter a fresh supply (500 ml) of distilled, deionized water through a clean 0.1-μm membrane filter. Filter 200 ml of this water through a 0.1-μm Nuclepore filter, prepare the electron microscope grid, and count exactly as in the procedures 8.1 - 8.4. Examine 20 grid squares and record this number of fibers. A maximum of two fibers in 20 grid squares is acceptable for the blank sample.

NOTE 1: The monitoring of the background level of asbestos is an integral part of the procedure. Upon initiating asbestos analytical work, blank samples must be run to establish the initial suitability of the laboratory environment, cleaning procedures, and reagents for carrying out asbestos analyses. Analytical determinations of asbestos can be carried out only after an acceptably low level of contamination has been established.

9. Calculations

9.1 Fiber Concentrations

Grid Square Counting Method - If the Grid Square Method of counting is employed, use the following formula to calculate the total asbestos fiber concentration in MFL.

$$C = (\overline{F} \times A_f)/(A_g \times V_o \times 1000)$$

If ashing is involved use the same formula but substituting the effective filtration area of the 25-mm diameter filter for A_f instead of that for the 47-mm diameter filter. If one-half the filter is ashed, multiple C by two.

C = Fiber concentration (MFL)

$\overline{F}$ = average number of fibers per grid opening

A_f = Effective filtration area of filter paper (mm^2) used in grid preparation for fiber counting

A_g = Average area of one grid square (mm^2)

V_o = Original volume of sample filtered (ml)

Field-of-View Counting Method - If the Field-of-View Method of counting is employed use the following formula to calculate the total asbestos fiber concentrations (MFL)

$$C = (\overline{F} \times A_f \times 1000)/(A_v \times V_o)$$

If ashing is involved use the same formula but substituting the effective filtration area of the 25-mm diameter filter for A_f instead of that for the 47-mm diameter filter.

C = Fiber concentration

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$\bar{F}$ = Average number of fibers per field of view

A_f = Effective filtration area of filter paper (mm^2) used in grid preparation for fiber counting

A_v = Area of one field of view (μm^2)

V_o = Original volume of sample filtered (ml)

9.2 Estimated Mass Concentration

Calculate the mass (µg) of each fiber counted using the following formula:

$$M = L \times W^2 \times D \times 10^{-6}$$

If the fiber content is predominantly chrysotile, the following formula may be used:

$$M = \frac{\pi}{4} \times L \times W^2 \times D \times 10^{-6}$$

where M = mass (µg)

L = length (µm)

W = width (µm)

D = density of fibers (g/cm^3)

Then calculate the mass concentration (µg/l) employing the following formula.

$$M_c = C \times \bar{M}_f \times 10$$

where M_c = mass concentration (µg/l)

C = fiber concentration (MFL)

$\bar{M}_f$ = mean mass per fiber (µg)

To calculate $\bar{M}_f$ use the following formula:

$$\bar{M}_f = \sum_{i=1}^{n} M_i / n$$

where M_i = mass of each fiber, respectively

n = number of fibers counted

NOTE 1: Because many of the amphibole fibers are lath shaped rather than square in cross section the computed mass will tend to be high since laths will in general tend to lie flat rather than on edge.

NOTE 2: Assume the following densities: Chrysotile 2.5, Amphibole 3.25

9.3 Aspect Ratio

The aspect ratio for each fiber is calculated by dividing the length by the width.

10. Reporting

10.1 Report the following concentration as MFL

a. Total fibers

b. Chrysotile

c. Amphibole

10.2 Use two significant figures for concentrations greater than 1 MFL, and one significant figure for concentrations less than 1 MFL.

10.3 Tabulate the size distribution, length and width.

10.4 Tabulate the aspect ratio distribution.

10.5 Report the calculated mass as µg/l.

10.6 Indicate the detection limit in MFL.

10.7 Indicate if less than five fibers were counted.

10.8 Include remarks concerning pertinent observations, (clumping, amount of organic matter, debris) amount of suspected though not identifiable as asbestos fibers (ambiguous).

11. Precision

11.1 Intra-Laboratory

The precision that is obtained within an individual laboratory is dependent upon the number of fibers counted. If 100 fibers are counted and the loading is at least 3.5 fibers/grid square, computer modeling of the counting procedure shows a relative standard deviation of about 10% can be expected.

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In actual practice some degradation from this precision will be observed but should not exceed ± 15% if several grids are prepared from the same filtered sample. The relative standard deviation of analyses of the same water sample in the same laboratory will increase due to sample preparation errors and a relative standard deviation of about about ± 25 - 35% will occur. As the number of fibers counted decreases, the precision will also decrease approximately proportional to $\sqrt{N}$ where N is the number of fibers counted.

Based upon the analysis of one laboratory utilizing a different analyst for each of three water samples, intra-laboratory precision data is presented in Table 1.

Table 1. Intra-Laboratory Precision

Sample Type	Number of Sample Aliquots Analyzed	Mean Fiber Concentration MFL (millions of asbestos fibers/l)	Precision, Relative Standard Deviation
Chrysotile (UICC)	26	23	37%
Crocidolite (UICC)	20	8	36%
Taconite (raw water)	20	16	24%

11.2 Inter-Laboratory

Based upon the analysis by various government and private industrial laboratories of filters prepared from nine water samples, inter-laboratory precision data of the method is presented in Table 2.

Table 2. Inter-Laboratory Precision

Sample Type	Number of Labs Reporting	Mean Fiber Concentration MFL (millions of asbestos fibers/l)	Precision, Relative Standard Deviation
Chrysotile	10	877	35%
"	9	119	43%
"	11	59	41%
"	9	31	65%
"	9	28	32%
"	3	25	35%
Amphibole	11	139	50%
"	4	95	52%
"	14	36	66%

12. Accuracy

12.1 Fiber Concentrations

As no standard reference materials are available, only approximate estimates of the accuracy of the procedure can be made. At 1 MFL, it is estimated that the results should be within a factor of 10 of the actual asbestos fiber content.

This method requires the positive identification of a fiber to be asbestos as a means for its quantitative determination. As the state-of-the-art precludes the positive identification of all of the asbestos fibers present, the results of this method, as expressed as MFL, will be biased on the low side and assuming no fiber loss represent 0.4 - 0.8 of the total asbestos fibers present.

12.2 Mass Concentrations

As in the case of the fiber concentrations, no standard samples of the size distribution found in water are available. The accuracy of the mass determination should be somewhat better than the fiber determination if a statistically significant number of the larger fibers, which contribute the major portion of the mass, are identified, measured, and counted. This will reduce the bias of low results due to difficulties in identification. At the same time, the assumption that the thickness of the fiber equals the width will result in a positive error in determining the volume of the fiber and thus give high results for the mass.

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13. Suggested Statistical Evaluation of Grid Fiber Counts

13.1 Since the fiber distribution on the sample filter, resulting from the method of filtration, has not been fully characterized, the fiber distribution obtained on the electron microscope grids for each sample should be tested statistically against an assumed distribution and a measure of the precision of the analysis should be provided.

13.2 Assume that the fibers are uniformly and randomly distributed on the sample filter and grids. One method for confirming this assumption is given below.

13.3 Using the chi-square test, determine whether the total number of fibers found in individual grid openings are randomly and uniformly distributed among the openings, by the following formula:

$$X^2 = \sum_{i=1}^{N} \frac{(n_i - np_i)^2}{np_i}$$

where X^2 = chi-square statistic

N = number of grid openings examined for the sample

n_i = total number of fibers found in each respective grid opening

n = total number of fibers found in N grid openings

p_i = ratio of the area of each respective grid opening to the sum of the areas of all grid openings examined

NOTE 1: If an average area for the grid squares has been measured as outlined in 8.4.5, the term np_i represents the mean fiber count per grid square.

If the value for X^2 exceeds the value listed in statistical tables for the 0.1% significance level with N-1 degrees of freedom, the fibers are not considered to be uniformly and randomly distributed among the grid openings. In this case, it is advisable to try to improve the uniformity of fiber deposition by filtering another aliquot of the sample and repeating the analysis.

13.4 If uniformity and randomness of fiber deposition on the microscope grids has been demonstrated as in 13.3, the 95% confidence interval about the mean fiber counts for chrysotile, amphibole, and total asbestos fibers may be determined using the following formulae:

$$(1)\qquad S_c = \left[\frac{N\sum_{i=1}^{N} X_i^2 - \left(\sum_{i=1}^{N} X_i\right)^2}{N(N-1)}\right]^{1/2}$$

where S_C = standard deviation of the chrysotile fiber count

N = number of grid openings examined for the sample

X_i = number of chrysotile fibers in each grid opening, respectively

Obtain the standard deviations of the fiber counts for amphibole asbestos fibers and for total asbestos fibers by substituting the corresponding value of X into equation (1).

$$(2)\qquad X_u = \bar{X} + \frac{tS_c}{\sqrt{N}}$$

$$(3)\qquad X_L = \bar{X} - \frac{tS_c}{\sqrt{N}}$$

where X_u = upper value of 95% confidence interval for chrysotile

X_L = lower value of 95% confidence interval for chrysotile

$\bar{X}$ = average number of fibers per grid opening

t = value listed in t-distribution tables at the 95% confidence level for a two tailed distribution with N-1 degree of freedom

S_c = standard deviation of the fiber counts for chrysotile

N = number of grid openings examined for the sample

The values of X_u and X_L can be converted to concentrations in millions of fibers per liter using the formula in section 9 and substituting either X_u or X_L for the term $\bar{F}$.

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Obtain the upper and lower values of the 95% confidence interval for amphibole asbestos fibers and total asbestos fibers by substituting the corresponding values of $\bar{X}$ and S into equations (2) and (3).

Report the precision of the analysis, in terms of the upper and lower limits of the 95% confidence interval, for chrysotile, amphibole, and total asbestos fiber content. If a lower limit is found to be negative, report the value of the limit as zero.

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